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MINING PRACTICE AT RAY MINES, NEVADA CONSOLIDATED COPPER CO., RAY, ARIZONA

BY

ROBERT W. THOMAS

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MINING PRACTICE AT RAY MINES, NEVADA CONSOLIDATED COPPER CO., RAY, ARIZ.[1]

By Robert W. Thomas[2]

INTRODUCTION

This paper describing the mining practices at the Ray mines at Ray, Arizona, is one of a series being prepared by the Bureau of Mines on mining practices, method and costs in the various mining districts of the United States.

At the present time (1929) all mineral production of the Ray district comes from the Ray Mines of the Nevada Consolidated Copper Co. The ore body is being worked from two shafts, designated as No. 1 and No. 2.

An undercut block-caving method of mining is used, variations of which are employed to fit different conditions. The present method has been developed by making improvements as experience was gained or when conditions changed. Refinements in practice are continually being made.

The Ray mine was the first in the Southwest to use a caving system of mining on a large scale.

ACKNOWLEDGMENTS

The author acknowledges the assistance of Moses Brown, chief engineer, and Dean La Grange, assistant chief engineer, Ray Mines, Nevada Consolidated Copper Co.; also that of E. D. Gardner, supervising engineer, and C. H. Johnson, assistant mining engineer, U. S. Bureau of Mines, Tucson, Ariz.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6167."

2 - One of the consulting engineers, U. S. Bureau of Mines.

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HISTORY OF DISTRICT

The Ray Mines of the Nevada Consolidated Copper Co. are situated in the southeast corner of Pinal County, Ariz., in the Mineral Creek mining district. The town of Ray is in the valley of Mineral Creek, a small stream which flows into the Gila River 6 miles below Ray. Figure 1 is a general map of the Ray district, showing topography, town sites, roads, and railroads. Ray is 90 miles from Phoenix via the Southern Pacific and the Ray & Gila Valley railroads, and approximately 85 miles by automobile roads by way of Florence and Superior. The estimated population of the Ray district is between 5,000 and 6,000 persons.

The ore is shipped by rail a distance of 20 miles to the company concentrator at Hayden. The concentrator was situated at Hayden, on the Gila River, in order to obtain an adequate supply of water for milling and sufficient area for tailings disposal. The concentrates are treated at Hayden at the smelter of the American Smelting and Refining Co.

Records indicate that mining for gold was in progress in the district as early as 1880. The gold mining operations were unsuccessful but undoubtedly led the discovery of copper.

In 1883 the Ray Copper Co. was organized and built a smelter at Ray to handle ores containing native copper. This organization, however, had only a short life, and there was no further activity in the district until about 1903 when an English company, the Ray Copper Mines, Ltd., was organized. This company established its general offices, built dwellings for its staff, and erected a 250-ton mill and smelter at Kelvin, Ariz., 5 miles below Ray on Mineral Creek. The mill at Kelvin was connected to the mine at Ray by a narrow-gage railroad. The mining operations of this company did not advance beyond the development stage, the work consisting of sinking a number of shafts and doing considerable drifting therefrom. Operations were carried on for approximately one year, when the property was shut down. Apparently this shut-down was brought about by the fact that the copper content of the ore--about 2 per cent--did not come up to the expected assay, and it was impossible at that time to operate successfully on an ore of such a low grade.

Mining activities were again resumed during the years 1906 and 1907, and several companies were organized, including the Ray Consolidated Copper Co., the Gila Copper Co., the Arizona Hercules Copper Co., and the Kelvin-Calumet Co. The Ray Consolidated Copper Co. and the Gila Copper Co. were soon consolidated, and in 1912 the Ray Consolidated Copper Co. also absorbed the Ray Central Mining Co. In 1926 the Ray Consolidated Copper Co. was taken over by the Nevada Consolidated Copper Co., and in 1927 the property of the Arizona Hercules Copper Co. was purchased by the Nevada Consolidated Copper Co.

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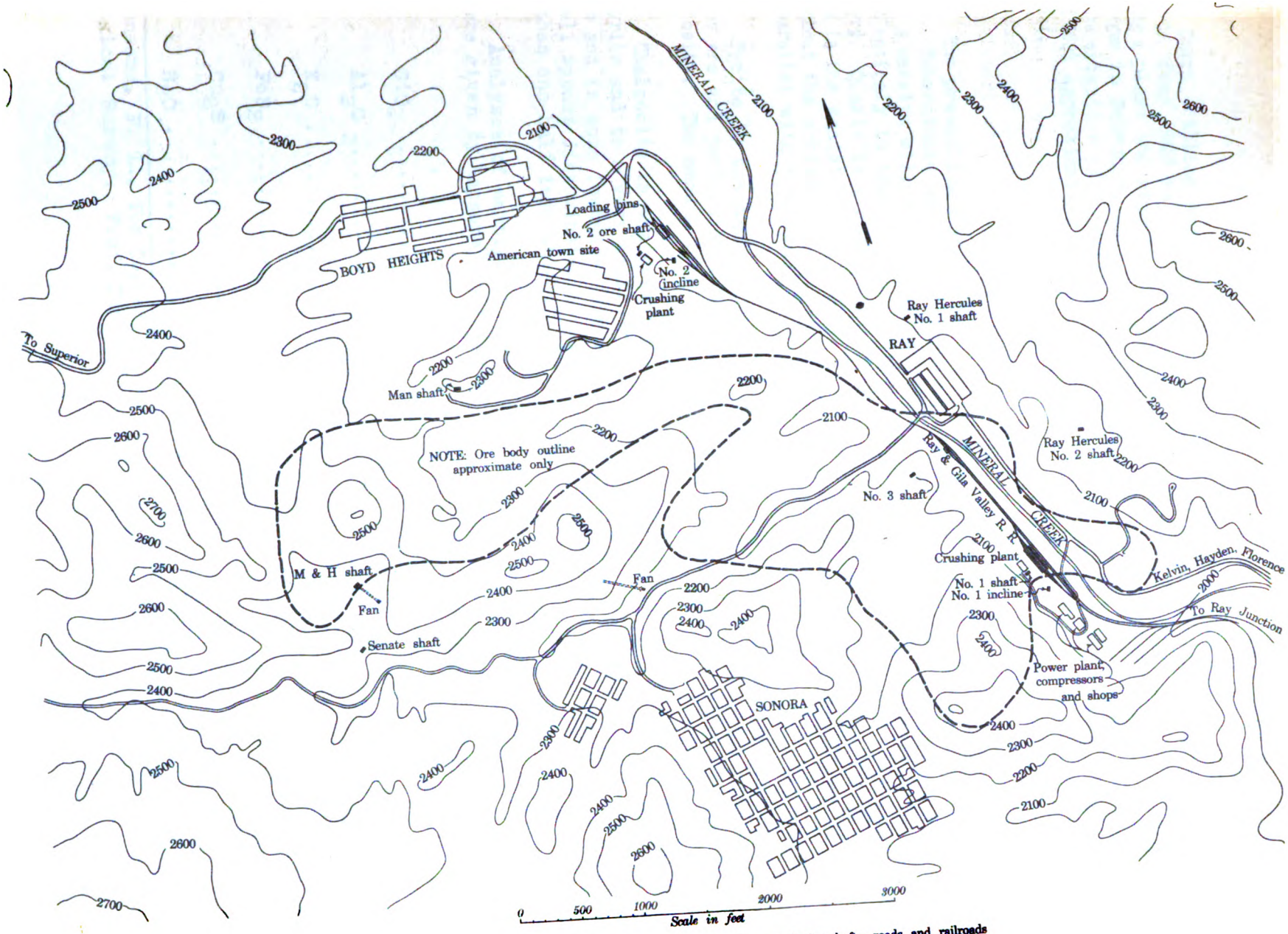

Figure 1.– Map of Ray district, showing topography, town sites, ore body, shafts, roads and railroads

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Churn drilling carried on by the Ray Consolidated Copper Co. and the Gila Copper Co. from 1907 to 1909 proved the existence of sufficient ore to warrant building a plant and starting mining operations. Shipments to the concentrator were begun in February, 1911, and operations have been continuous since that date, with the exception of a period from April 1921, to April, 1922. From 1911 to the end of 1928 approximately 1,003,000,000 pounds of copper was produced in the Ray district.

GEOLOGY

The prevailing rock in the mineralized area of Ray is a quartz-sericite schist. Subordinate amounts of porphyry intrusives occur mainly outside of the ore zone. A massive sill of diabase underlies part of the ore body. The ore body is almost entirely in the schist, a relatively small amount of ore occurring in granite porphyry. Quartz is a microscopic primary constituent of the rock. Secondary quartz occurs in the schist and as gangue in stringers of chalcocite. Sericite occurs throughout the schist, and in places is abundant along some of the planes of weakness parallel with the schistosity.

Pyrite is the most abundant sulphide mineral and occurs disseminated throughout the ore and particularly along the lines of fracturing and in the planes of schistosity. The amount of pyrite varies in different parts of the deposit.

Chalcocite is the essential ore mineral and has been deposited by replacing the pyrite and to a very small extend the chalcopyrite. It is associated with the pyrite and in some instances has entirely replaced it. It occurs characteristically as small specks disseminated throughout the ore and as distinct veinlets generally less than one-half inch thick.

Analyses of samples of schist ore from the No. 2 and the No. 1 shaft workings are given by Ransome[3] as follows:

	No. 2 mine	No. 1 mine
SiO_2	69.35	68.52
Al_2O_3	12.96	15.35
K_2O	5.02	5.77
FeS_2	4.98	2.03
Cu_2S	2.85	3.13
H_2O	2.38	2.27

3 - Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz.: United States Geological Survey. Prof. Paper 115, 1919, p. 159.

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A system of major faults with considerable fault breccia extends northwestward and southeastward through the ore body. Another major fault running at right angles to the main system occurs near the west end of the ore body. A number of minor faults also are present and gouge-filled fissures are common. The faults and fissures that cut through the capping tend to prevent doming or arching of the overburden when the stopes are drawn.

Several sets of planes of weakness occur in the schist, the most pronoun one generally being with the schistosity. A second set less well defined runs a right angles to the schistosity and parallel to its strike. Other sets of minor planes of weakness run in different directions. Due to the weakness along the p of schistosity the ore generally has a tendency to break down in slabs. These f pieces next break along planes of weakness into rectangular shapes. The schistosity has no uniform strike or dip.

The ore in the mine generally breaks coarsely; in some places, however, breaks up quite readily. The ore as it falls in the caving blocks is in larger fragments than at most mines using a caving system of mining. The hardest groun to break at Ray contains a large proportion of secondary quartz that has penetrat the whole rock structure rather than being confined to stringers. There may also be recementation of the planes of weakness by the quartz which, in mining, has a further tendency to cause the rock to break into large fragments. This is particularly true in one area where there appears to be a center of fissuring and dyk

Those parts of the ore body having large proportions of sericite cave ver readily, but this ore is likely to pack if not drawn immediately and continuously after undercutting. In this type of ground it is essential that preliminary development work be done well in advance of mining operations to allow the ground t dry out as much as possible.

On standing exposed in workings the ore usually slacks, with resultant sloughing from the top and sides; this makes timbering necessary in all permanent workings in the ore.

In some places the ore is limited horizontally by faults which undoubtedl have acted as dams to enriching solutions, but in most instances the values fade out into the surrounding rock and the economic ore limits are obtained by assay.

The bottom of the ore body is sometimes limited by a diabase sheet or by flat-lying faults. In most places, however, it is limited by the diminution in grade.

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A sharp line of demarcation generally occurs between the ore and the leached iron-stained capping. Parts of the capping in which secondary quartz is abundant have a tendency to arch over the areas drawn underneath and break into large blocks. Other parts of the capping are soft and friable and run readily once the rock is disturbed. In such areas care must be taken to prevent pipes of waste coming through the ore during drawing operations, particularly if the ore has settled and become packed. Packing is more likely to occur when the material is damp. This finely divided capping is also likely to filter down through the coarser ore as the drawing nears completion.

The ore body is irregular both in plan and section. It occurs as two connected flat lobes with 60 per cent of the ore reserves in one and 40 per cent in the other. The No. 2 shaft workings are in the larger lobe and the No. 1 in the smaller. The main axis of the ore body is about 7,000 feet long and strikes nearly east and west. The average width of the ore body is about 1,500 feet. It is from 40 to 400 feet thick and averages about 120 feet. It is undulating but has a general dip of about 100 feet vertically to 1,000 feet horizontally. The thickness of the capping is between 40 and 600 feet and averages about 250 feet.

PROSPECTING AND EXPLORATION

The first prospecting of the Ray ore body was done by sinking shafts and drifting from them. The next prospecting was carried on by churn drilling, holes being drilled over the entire area at the corners of 200-foot squares whenever practicable. Further exploration has been carried on underground by diamond drilling, "long-hole" drilling, and through the usual procedure of drifting and raising. This last has been confined chiefly to determining extensions of the ore body outside of the limits indicated by the original churn drilling.

At present "long-hole" drilling has almost entirely replaced prospect raising. The drill used for this purpose is a Leyner-type machine with air-motor rotation and is mounted on a 42-inch cradle. It is fitted with a special water connection in front of the chuck. Two vertical columns and a cross-arm are used for the set-up, which requires a station at least 5 by 7 feet in the clear. One and one-quarter inch round hollow steel is used in 3, 6, and 9 foot lengths, jointed by threaded sleeve couplings, the shorter lengths being used only to make changes. The bits are detachable and are of the usual cross form with a 5 by 14° double taper. The largest-sized bit is 3-1/2 inches in diameter, and each reduction in gauge is one-eighth inch. Holes up to 90 feet long are drilled usually on a 45° pitch. The pitch may be as great as 55°, and flat or downward holes can be drilled, though less readily, dependent upon the nature of the ground. A pump is used for forcing water through the hollow steel when a sufficient head of mine water is not available.

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The average footage drilled per machine per shift over a long period has been 23 feet; this included moving and setting up.

The sample interval is 6 feet. Sludge is split at the collar of the hole by an automatic sampler, devised locally for this purpose. The sample is dried over a heater outside the mine before being sent to the assay office.

SAMPLING AND ESTIMATION

In churn drilling, samples were obtained by the use of a split divider. A careful record was kept of the type of material being drilled through, the color of the sludge and the character of its various mineral constituents, the weight of material cut for each 5 feet of drilling, the size of bit, and the length and size of the casing in the hole. From the weight of the sample and the size of the bit it was possible to determine whether there was caving in the hole and to thus arrive at some conclusion as to the accuracy of each 5-foot sample. The samples were assayed locally by the iodide method, and the remainder of the pulp was sent away for determination of the copper by the electrolytic method.

In the case of diamond drilling, all the sludge and water from each 5 feet of drilling was retained until the sludge had completely settled, allowed decanting of the clear water. The core was also retained. The percentage of core, however, was so small, being approximately 5 per cent of the total, that it was not used in connection with the final assay results.

Every precaution was used to obtain fair samples from the diamond-drill holes. If upon lowering the rods into the hole after a 5-foot interval was drilled it was found that the bit did not return to its former position by 1 foot, the operators were required to remove the rods and case the hole to the bottom. All samples were assayed and records kept as in churn drilling.

In considering the ore body as a whole, the probable tonnage and grade of the ore in place is calculated entirely from the results of churn and diamond drilling, except in the case of a few isolated ore extensions. The first step in calculating the tonnage and grade of the ore body was to make vertical sections along the north-south lines of drill holes, showing the ore outline and the assays of each hole. Next the ore area and average grade of each section were calculated. Finally the volume and average grade of ore between each pair of adjacent sections were calculated and these in turn combined to give a volume and grade for the whole ore body. A factor of 12-1/2 cubic feet per ton was used to calculate tonnages; this figure was determined by actual tests on the ore. The ore was not assumed to extend beyond the drill holes until shown to do so by subsequent development.

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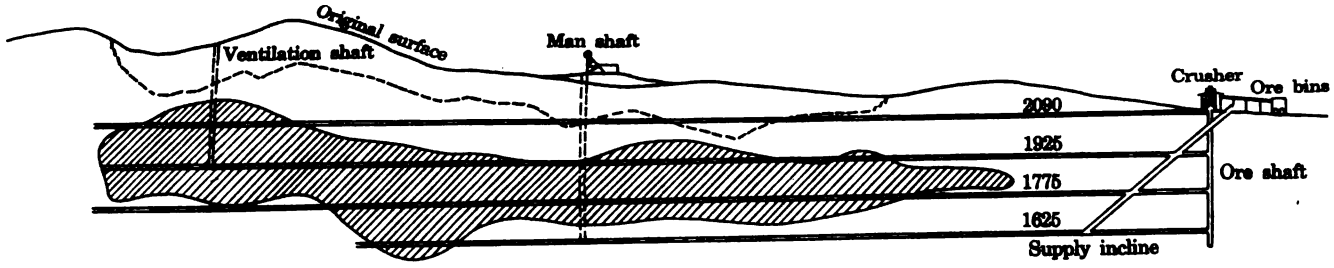
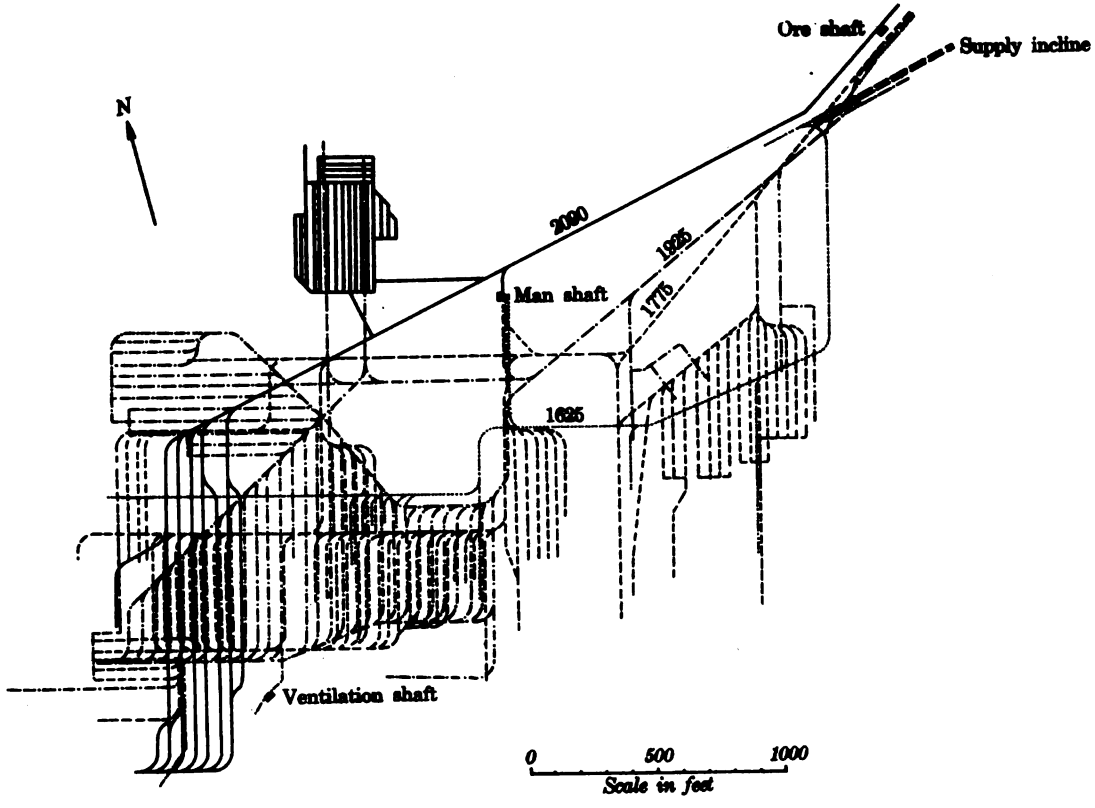


SW.–NE. section
Same scale as plan



Plan

Figure 2.– Plan and section of No. 2 mine

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In the course of underground mining operations, calculations are made of the tonnage and grade of the individual blocks being mined; the necessary information for such calculations is obtained from actual stope sampling and mine development. In developing a block of ground for mining, as will later be explained in detail, cut-off shrinkage stopes are carried up to the capping on four sides of the block. Grab samples are taken at 10-foot vertical intervals from the broken ore across the width of the stope and for a length of 12-1/2 feet along the stope. The original method of sampling stopes was to cut channels, but owing to the irregularity of the stopes it was very difficult to obtain satisfactory samples in this manner, and grab sampling as described above has been adopted. The assays of samples taken from a block during drawing are adjusted to the mine head which is obtained from an automatic mechanical sampler.

In development work, drifts and raises are sampled by both grab samples and cut samples. The grab samples are taken to secure a daily record of the various headings. The cut samples are used for permanent records and are placed on assay maps and sections of a scale 50 feet to the inch. Cut samples are taken in drifts at 12-1/2-foot intervals on alternate sides of the drifts. Every care is exercised to obtain as uniform a sample as possible. In cases where the planes of schistosity are well defined, the samples are taken at right angles to these planes.

In determining the tonnage and assay value of a mining block all assays which are available within the block are considered. As the cut-off stopes are carried to the capping all around the block and are sampled, as previously explained, they provide a series of vertical sections from which the tonnage and copper contents of the block can be computed. Additional data are also provided by the cut sampling of development drifts which may have been previously driven to explore the country or to provide access for subsequent mining; and these results are also taken into consideration in arriving at a final estimate of probable assay value of the ore within the block.

The future development program and the sequence of subsequent mining operations are controlled by a consideration of the assay maps and sections, upon which are also shown the geological features.

METHODS OF DEVELOPMENT AND MINING

Development

The development of the mine as a whole has been by two vertical shafts for the hoisting of ore, two inclined shafts provided with stairways for the handling of supplies and mine waste and for emergency exits, and one vertical shaft for the handling of men (figs. 1 and 2). The main haulage levels have been established at approximately 150-foot intervals, and to date four of these have been developed. The mining of the ore from various levels was made necessary by the shape of the ore body, which dips from the west to the east.

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Shafts. Hoisting and man shafts and supply inclines are concreted. This reduces maintenance costs, insures 24-hour-a-day operation, decreases fire hazards, and increases ventilation efficiency.

Figures 3 and 4 are plans and sections of a main hoisting shaft and an inclined supply shaft.

All ventilation shafts, both intake and exhaust, are also concreted and are equipped with steel ladderways as emergency exits.

The two compartment Senate shaft to be used for ventilation was recently sunk from the surface to a depth of 482 feet.

The ordinary V-cut round with 10 to 16 holes 5 feet deep was used. Thirty per cent strength ammonia dynamite and electric delay blasting caps were used to blast the round. In firm ground the shaft was timbered with 8 by 8 inch timber and the usual six-post shaft set. In softer ground 10 by 10 inch timber was used. Work was done by a crew of three men in the bottom of the shaft. Concreting was done from the bottom up. Dividers and posts were removed and concrete poured around the remaining timber, sufficient lagging being removed to allow the concrete to come into contact with the solid rock at the sides. The original sets were of such size as to allow the reinforcing to be placed in front of the timber.

Drifts. Main haulage drifts are broken 8 by 9 feet and haulage laterals 8 by 8-1/2 feet in cross section. Two crews of two men each are given contracts for running drifts. A crew consists of a miner and a helper. One crew works on day shift and the other on afternoon shift, and each drills, shoots, or mucks out the round or does the timbering, as the case may be. A round is generally drilled on one shift and mucked out on the next, and the timbering is done by the shift following.

Drifter machines, supported by horizontal bars, are generally used for drilling the rounds; vertical bars are used occasionally.

Each contractor uses his own choice of round; a V-cut round with from 13 to 15 holes is commonly employed. Rounds are blasted with 30 per cent strength ammonia dynamite and No. 6 blasting caps. Five feet, the distance for a set, is usually broken by each round.

Rock broken in drifting is shoveled by hand into 5-ton cars, which are then handled by the haulage locomotive.

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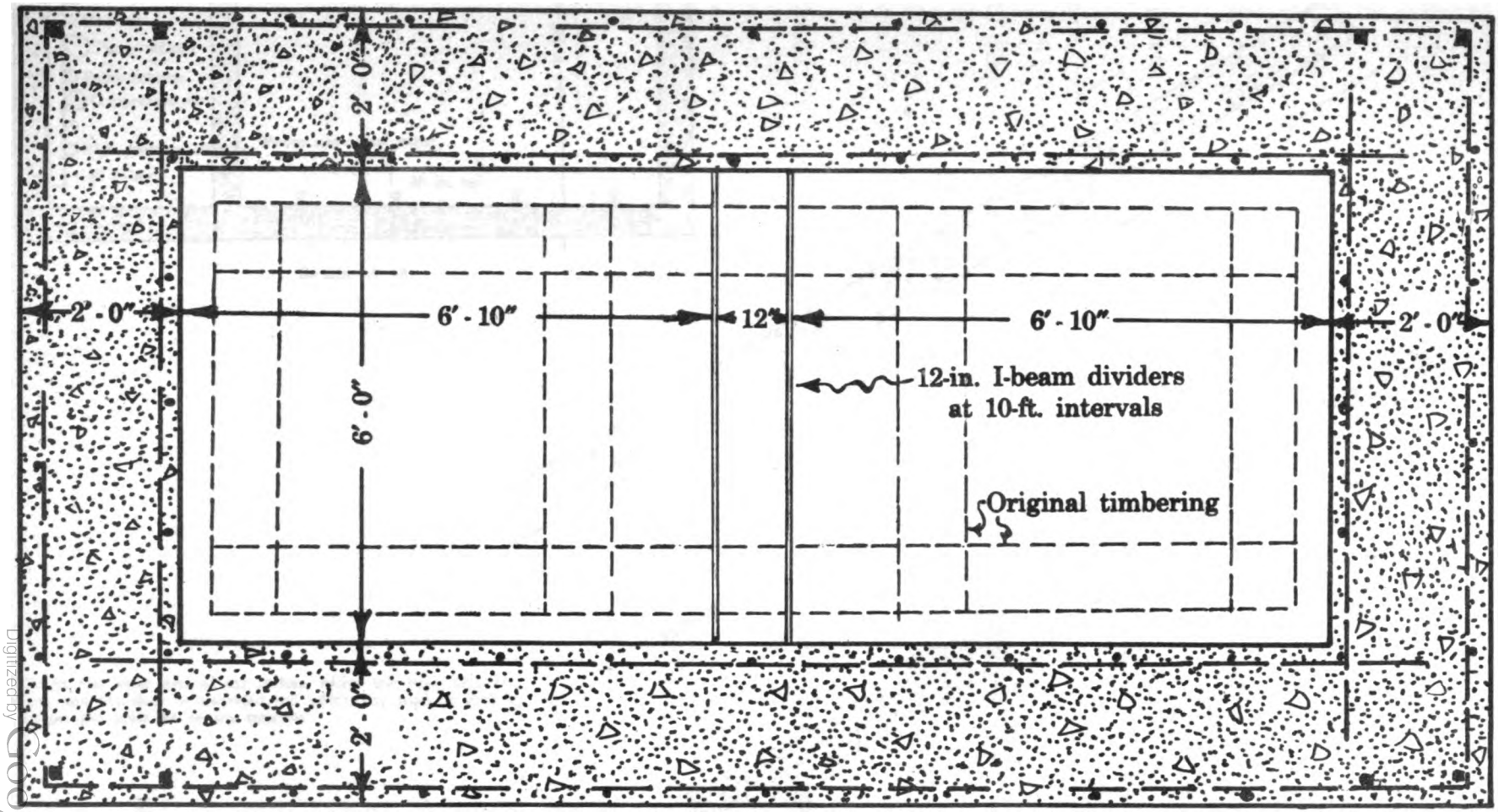

Figure 3. — Section of main ore hoisting shaft

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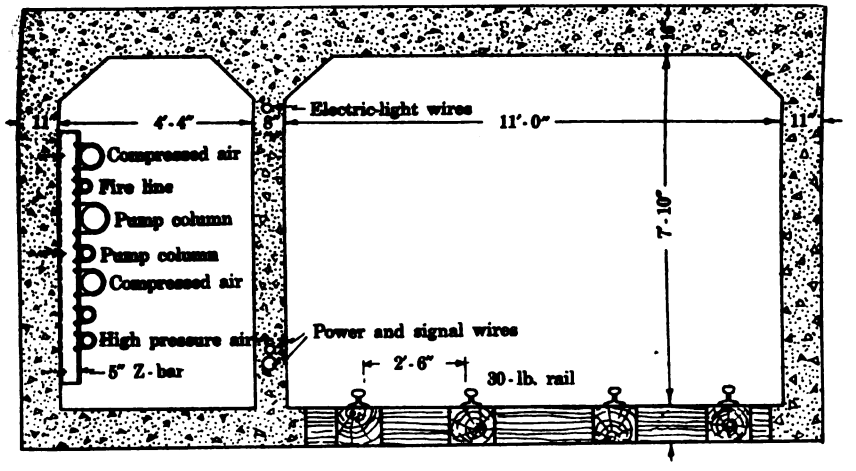
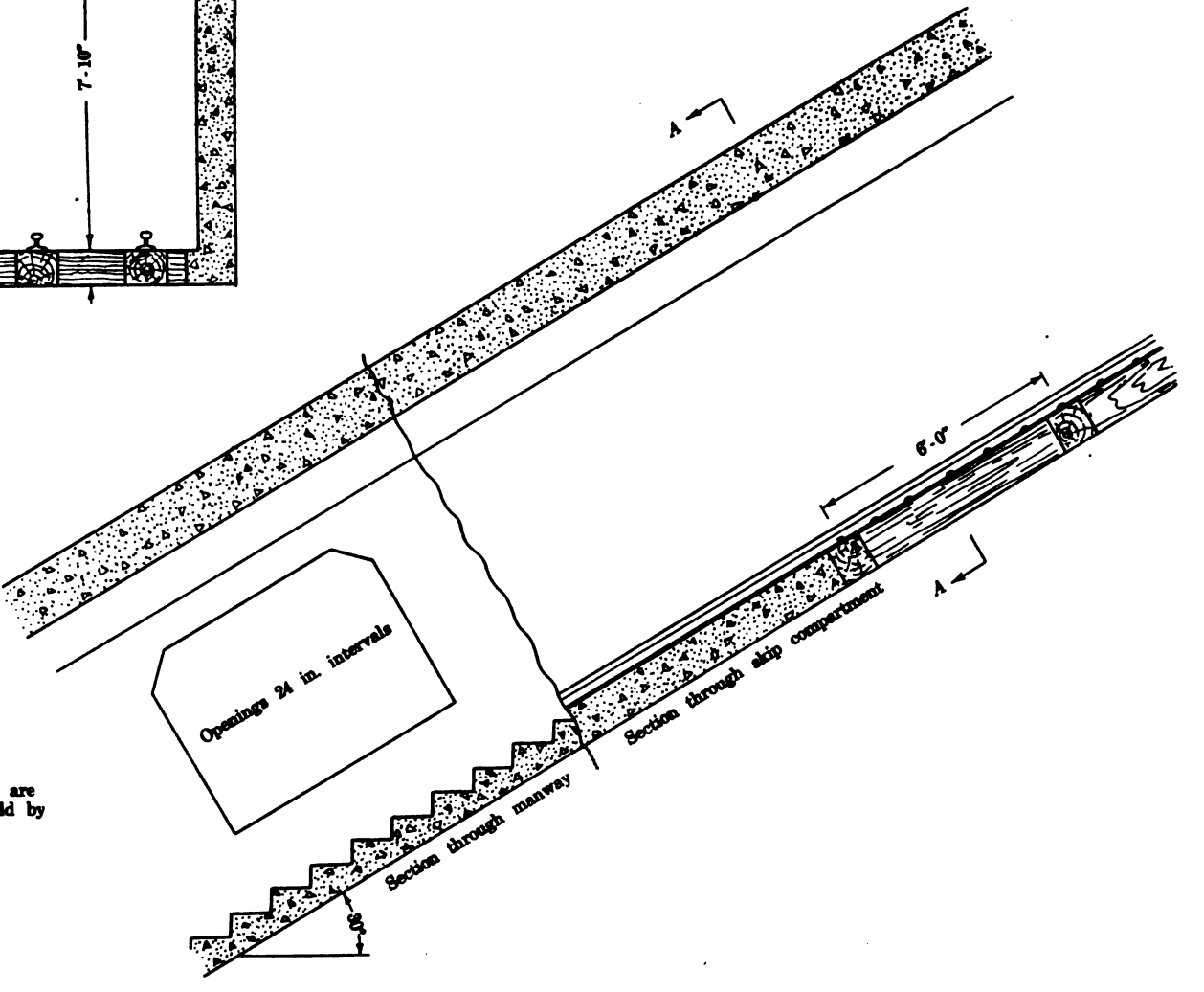


Section *A*–*A*



NOTE: In the lower part of the incline, where the track sills are omitted, and the floor is concreted full width, the track is held by tie straps and bolts set in the concrete

Figure 4.– Plan and section of inclined supply shaft

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Drift sets consists of 12 by 12 inch posts and 12 by 14 inch caps. The main haulage sets are 7 by 8 feet and the laterals 7 by 7-1/2 feet in the clear.

Timbering usually is kept close to the face, as the ground is soft and sloughs on exposure. In fact, spiling must be used overhead in drifting through the softer parts of the formation. Formerly drift sets were set on 6 by 12 sills. However, it was found that in swelling ground the sills were forced upward in the floor of the drift, carrying the track with them. Present practice, therefore, consists of standing the posts on 6 by 12 inch foot blocks and using what are known as batter blocks to resist side pressure (see fig. 5).

Peg lagging is used in some swelling ground. The lagging is kept from direct pressure upon the post by pegging it back with wooden pegs about one-half inch thick and 4 inches long. When pressure causes the pegs to break or split, the lagging can be removed and some ground taken out to relieve the pressure before it has become great enough to break the post.

In heavy ground on main haulage lines the drifts are concreted (see fig. 6).

Sublevel fringe drifts are timbered with 8 by 8 inch timbers with posts 7 feet 2 inches long, battered 2 inches to the foot. Sublevel draw laterals are run untimbered and are approximately 3-1/2 by 5 feet in cross section. These drifts are driven by contract. Rounds are drilled with hand-held plugger-type machines, the weight of which is supported by a ring which is slipped over the end of a piece of drill steel.

Winzes. To insure small and exact connections at the points where raises from the haulage level tap the grizzly level, short winzes are sunk in the floor of the sublevel drifts previous to raising. These winzes consist of one round drilled, blasted, and mucked out, plus a 5-foot hole in the center of the winze in which a piece of old steel or pipe is left. This is done because it is desired to have firm footing for the grizzly set and to take out as little ground as possible at this point. The raises may be driven first, but in this case they are stopped about 6 feet below the floor of the sublevel.

Raises. Most of the raising is done in connection with the sublevel method of mining and consists of storage raises between the haulage level and the sublevel. These are started from chutes in the pony sets above the drift timbers on the haulage level. If the ground permits, they are driven untimbered and about 4 by 4 feet in section. If timber is necessary, 6 by 8 inch cribbing is used, giving an inside dimension of 4 by 4 feet. As the raises are never over 30 or 35

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feet long, a manway compartment is not required. In untimbered raises the miners work on a platform supported on stulls. These raises are never driven flatter than 55° from the horizontal; at any pitch steeper than this the ore flows freely and does not build up on the bottom of the raise.

Mining Methods

Four variations of one general mining method are used at the Ray mine, as are designated as follows: Hand-tramming method, motor method, modified motor method, and sublevel method. These have been developed to apply to particular conditions in the ore body, such as thickness, distance above the haulage level, and other physical characteristics.

The hand-tramming method is used to mine small bodies of fringe ore which are too high above the main haulage level to permit the use of either the motor system or the modified motor system and which do not have sufficient vertical thickness to warrant the application of the sublevel system.

The motor system, in which the ore is drawn directly into cars on the haulage level, was extensively used in the past but is now confined to areas where the ore comes down to the haulage level and extends less than 100 feet above it and where the ground is hard and has no tendency to cement or pack after being caved.

The modified motor system, with its better spacing of draw points, is used in areas of softer ground and where the height of ore is less than 150 feet above the haulage level.

The sublevel system is the preferred method and is used in all types of ground where the thickness of ore above the haulage level is 150 feet or more. There is no essential difference in the methods described other than from a development standpoint. That is, after the preliminary development work has been completed, the actual mining of the ore is the same in all cases.

Development of Present Methods. The present system of mining is an improvement on the original method developed at Ray at the beginning of operations. The original method consisted essentially of alternate shrinkage stopes and pillars. All the stopes were carried through to the capping, after which the pillars were cut off at the base and caved.[4] In order that the present method may be compared with the original method a short description of the latter is given here. In a hand-tramming section, six to eight laterals were driven through a block on 25-

4 - Blackner, Lester A., Underground Mining Systems of Ray Consolidated Copper Co.: Trans. Am. Inst. of Min. Eng. vol. 52, 1916, pp. 381-423.

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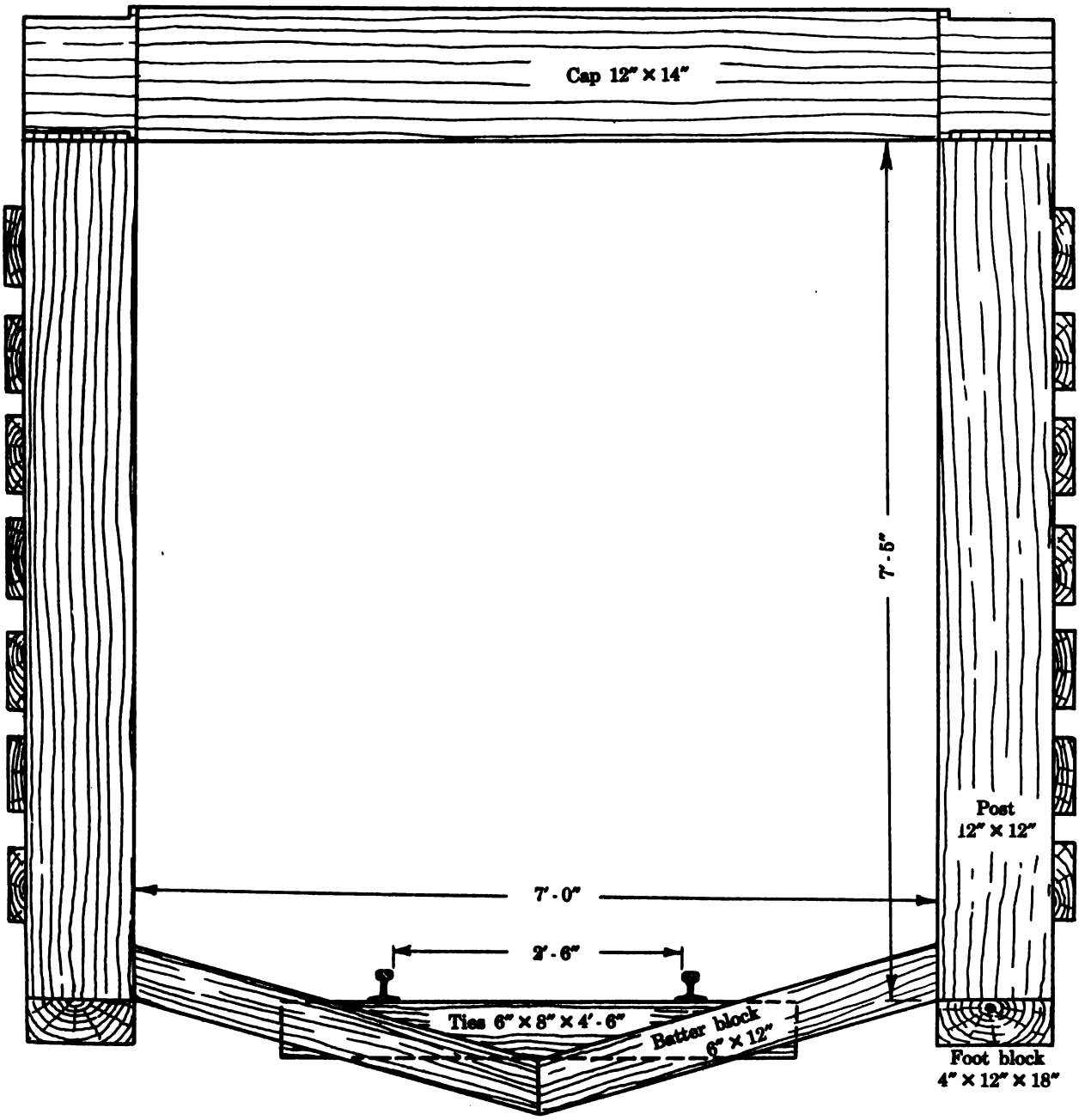

Figure 5. – Drift set, showing use of batter blocks

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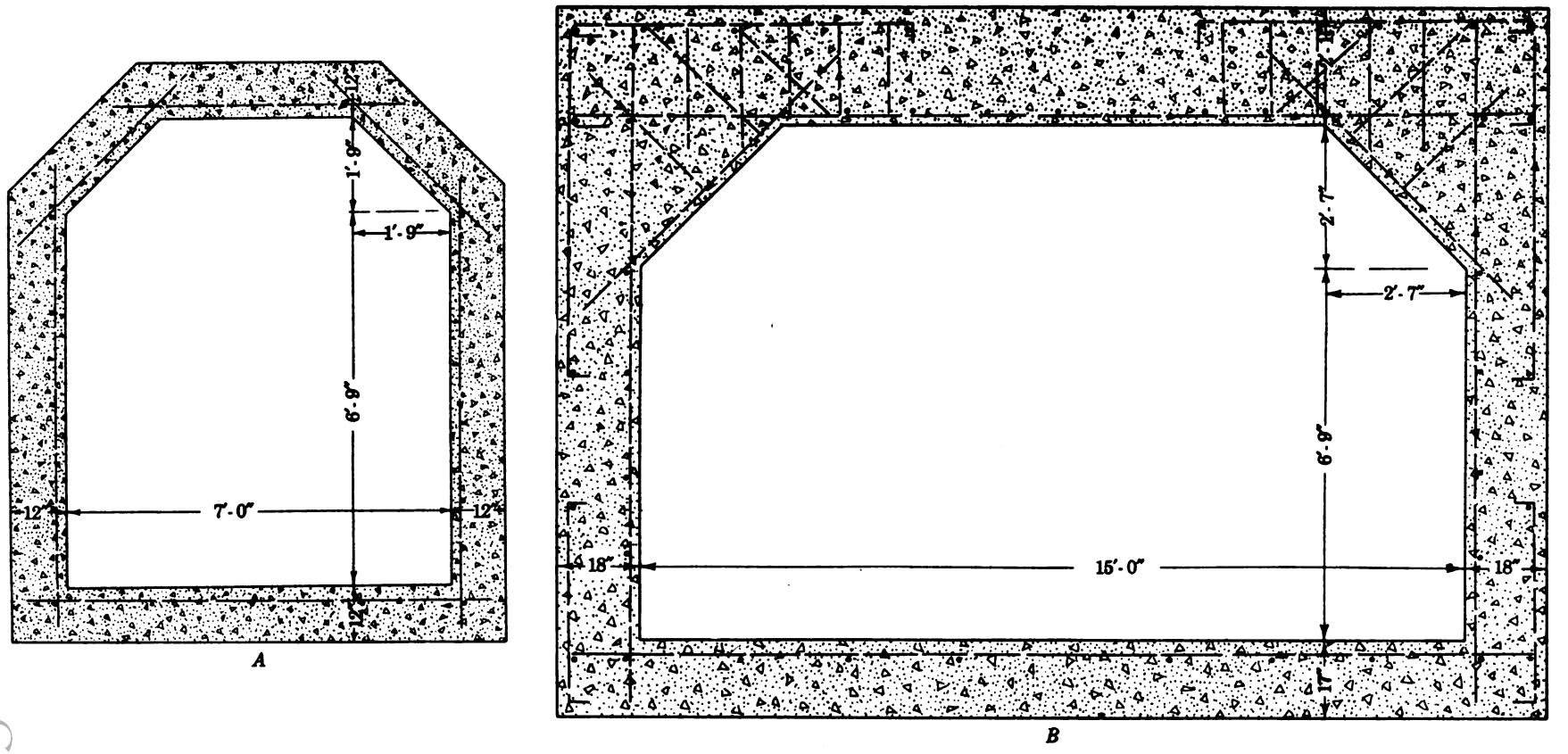

Figure 6.— Concreted drifts. *A*, Single track drift; *B*, double track drift

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foot centers (fig. 7). Shrinkage stopes on 25-foot centers and at right angles to the laterals were then carried up to the capping. These were made as wide as the ground would permit in order to keep the remaining pillars as small as possible. In starting the shrinkage stopes raises were put up on a 45° slope from both sides of the laterals until they intersected, forming the starting cut of the stope. The same method of starting undercut stopes is still used and is described in detail later.

As soon as the shrinkage stopes were completed the pillars between them were undercut. To effect this, raises were put up in the pillars in the same manner as in starting the shrinkage stopes. The pillar stopes were carried up to above the tops of the cones made by the intersecting raises between the drifts, after which the sides and back were drilled and blasted. This final round broke into the shrinkage stopes on either side, thus completely undercutting the pillar. The block was then ready for drawing. The ore was drawn into small cars and hand-trammed to transfer raises from which it was loaded into motor cars on the haulage level. The same procedure was followed in the motor method with the exception that the laterals were spaced on 50-foot instead of 25-foot centers and the ore was drawn directly into the motor cars, thus avoiding the expense of hand-tramming.

These systems gave satisfactory results in the hard character of ground met with on the upper mining levels, but in softer areas encountered as operations were continued on lower horizons the motor method gave less satisfactory results.

As the chimneying or piping of the softer ore and the wide spacing of the draw points resulted in a decreased recovery, to complete the extraction it was necessary to mine the ridge of ore remaining between the main haulage laterals. This was accomplished by driving hand-tramming drifts termed locally "cone drifts" between the main haulage laterals. These drifts were originally driven on the haulage sill but later were dropped to a point 10 feet below the haulage sill in order to place the drift in firmer ground and avoid excessive timber repairs.

After the driving of the cone drifts was completed, shrinkage stopes were carried over the top of these drifts in the center of the remaining ridge of ore. These stopes were carried to the broken ground, and when completed, drawing operations were accomplished by hand-tramming the ore from the chutes to transfer raises connecting with a lower haulage level. To improve this condition the motor haulage laterals were subsequently spaced at 40-foot centers instead of 50-foot centers, thus bringing the draw points closer together. This gave more satisfactory results

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with no apparent increase in the timber difficulties experienced in the use of the 50-foot center drifts. It was however, necessary in some areas of soft and cement ground to put in the cone drifts between the 40-foot center laterals as described in the case of the 50-foot center laterals in order to secure a satisfactory recovery from the mining block.

To reduce the cost of mining and improve extraction, it was then decided t cave a block of ground by undercutting by the stope-and-pillar method as before, but with the stopes carried only to a point approximately 8 feet above the cone connections and then blasted with a final flaring round, and with the pillars also attacked as before to complete the undercutting. Improved results were obtained by this method, and it was decided to use this system of undercutting wherever po ble

An objection to this method of procedure was that in certain instances th ground had a tendency to arch over above the elevation to which the undercut stop were carried, resulting in a loss of recovery through failure of the shoulder of t arch to cave. In order to prevent such an occurrence, it was subsequently decide to carry cut-off stopes all around the block up to capping, or to the level above and undercut the central mass of the block by the method previously described.

This method of mining, which is locally known as the motor method, was be extensively used at Ray Mines during the years 1925 and 1926,[5] though in recent years it has been in great measure replaced by the sublevel system.

Motor Method. The following description covers the development and minin of a motor stoping area in which the ore does not exceed a height of 100 feet abo the haulage level.

A fringe drift is driven along the side of the ore body and from it later are driven on 40-foot centers through the ore body or to a point where less than 50 feet of ore extends above the haulage level. These drifts and laterals are 7 feet in the clear inside the timbers. Posts are 7 feet, 6 inches long, and caps 9 feet over-all; 12 by 12 inch timber is used. Four laterals usually constitute a panel, the width of which is 160 feet; the length of a drawing block is usually 100 feet. The block on the far end of the panel is worked first; then the next i order retreating to the fringe drift. It is not practical to work a whole panel as a unit on account of the expense of keeping the laterals open. As soon as the laterals are completed the work of putting in pony sets is started, and preparati are made for mining the first block (see fig. 8). When the work of installing th

5 - Thomas, Robert W., Mining Methods Used at Ray Mines: The Mining Congress Jour- nal, Vol. 12, page 366, May, 1926.

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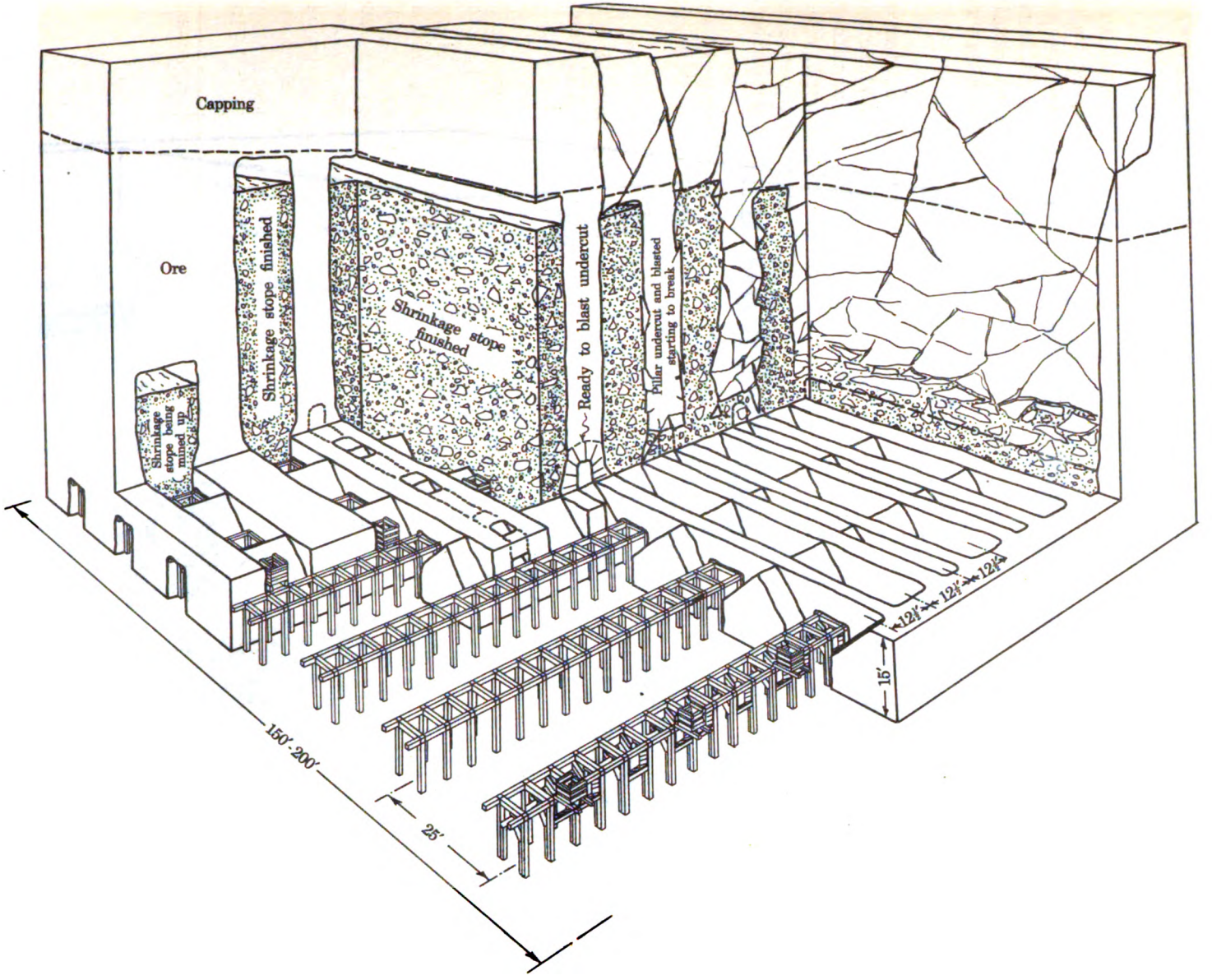

Figure 7.– Hand-tramming caving method

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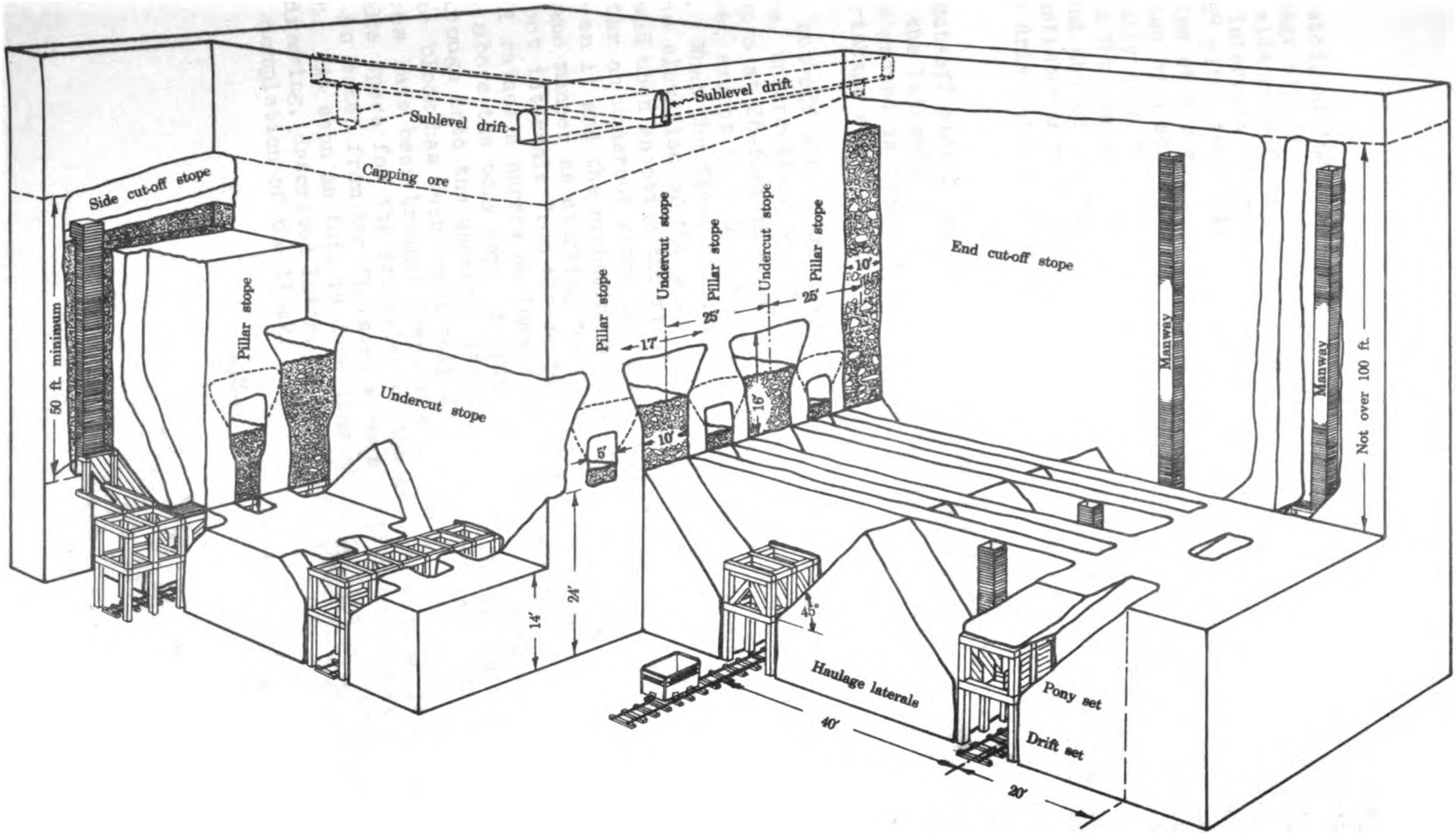

Figure 8.– Motor-haulage caving method

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pony sets is completed, mining operations are started. This work consists of first carrying up shrinkage stopes at the boundaries of the block, termed locally "cut-off stopes." The side cut-off stopes are begun by driving raises 10 feet wide, spaced at 25-foot intervals, from the two outside haulage laterals. These raises are driven on a 45° pitch to a point 20 feet outside of the center of the drift; that is, to the side of the block. As soon as the raises are completed they are connected at the top by a small plugger drift, and from this drift the stope is carried up vertically as a regular shrinkage stope. Cribbed manways are maintained in the ends and in the center of the stope by putting a partition in the 10-foot raise at either end and in one raise at the central position. Ventilation is provided for by an untimbered raise in the center of each stope. These raises connect with small drifts driven for ventilation purposes on an upper level near the top of the ore.

The end cut-off stopes are started by connecting raises 10 feet wide on a 45° pitch across the laterals at both ends of the block. As soon as the raises are completed the ground is broken down over the timber, and the stopes are carried up as regular shrinkage stopes.

When the cut-off stopes are completed to the capping, or to a previously mined level above, undercutting operations are started. This consists of carrying up shrinkage stopes at 25-foot intervals across the block exactly similar to the end cut-off stope, except that they are not carried so high. These are called undercut stopes. When the final round is completed these stopes are approximately 26 feet above the elevation of the pony cap. They are carried not less than 10 feet in width, and upon shooting the final round are flared out at the top to 17 or 18 feet. After an undercut stope is finished, work can be started undercutting the pillar between it and the preceding stope. Raises are connected across in the pillar in the same manner as starting the undercut stopes, but only 5 feet in width and at 12-1/2-foot intervals from the center lines of the undercut stopes. Out of the connected raises a narrow shrinkage stope is carried up to a point approximately 20 feet above the pony cap. A final round is then drilled and blasted which breaks through into the undercut stopes on both sides. This work is continued until the block has been completely undercut. After two undercut stopes and two pillar stopes have been brought down, a small amount of ore is usually drawn from them to make space for the expansion of the ground above as it caves. Only a limited draw is taken from the finished stopes until such time as the block is entirely undercut. As soon as this is done drawing operations are started, and the usual form of drawing, described later, is then maintained throughout the drawing area until the completion of the block.

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In this method of mining some difficulty is experienced because of the weight on the timbers, which necessitates timber repairs. Repair work, in this method, usually prevents the passage of trains and thus may stop drawing from an entire lateral. However, the comparatively small height of ore permits rapid drawing from the block, and generally drawing is completed before excessive timb repairs are necessary.

For sections of soft ground and where the height of ore exceeds 100 feet but is not sufficient to warrant the use of the sublevel system, the modified motor system has been developed.

Modified Motor Method. The modified motor method has the advantage of h ing some of the weight off of the motor-haulage laterals and thus minimizes dela and interruptions in train service. It also allows a closer spacing of the draw points, which is essential in softer ground.

The details of the modified motor method are outlined in Figure 9. Late are spaced 40 feet apart, as for the standard motor method, and similarly the pa are four drifts or 160 feet wide. The length of the block, however, is usually feet. The increased length of the block is permissible because less timber repa ing is required in the laterals. Pony sets are put in over the laterals for the entire length of the block, and chutes are placed in the pony sets at 15-foot in tervals. Chutes are staggered, so that those on the same side of the lateral ar 30 feet apart.

After the chutes are completed, raises are started and draw sets install (see fig. 10). These sets are supported by stringers running from the top of th pony cap into the solid ground at right angles to the lateral. Figure 11 is a plan of such a modified motor section, showing the location of the draw sets wit respect to the haulage laterals.

The stoping and undercutting operations in this system are similar to th of the standard motor system. Raises are put up from the outside draw sets of th outside laterals of the block. These raises, which are called "throat raises," inclined slightly so that when they reach a height of 7 feet above the top of th set they are 15 feet from center to center. From the tops of the throat raises, 45° inclined raises are next run at right angles to the lateral; the raises on th outside of the block are carried to a point 20 feet outside of the haulage drift These outside raises are then connected parallel to the haulage lateral drifts b a small drift or inclined raises to form the base of the side shrinkage stope. Manway entrances for these stopes are carried up from the ends of the draw sets. As in the standard motor method, cribbed manways are carried in each end and in center of the stope with an open raise in the center for ventilation.

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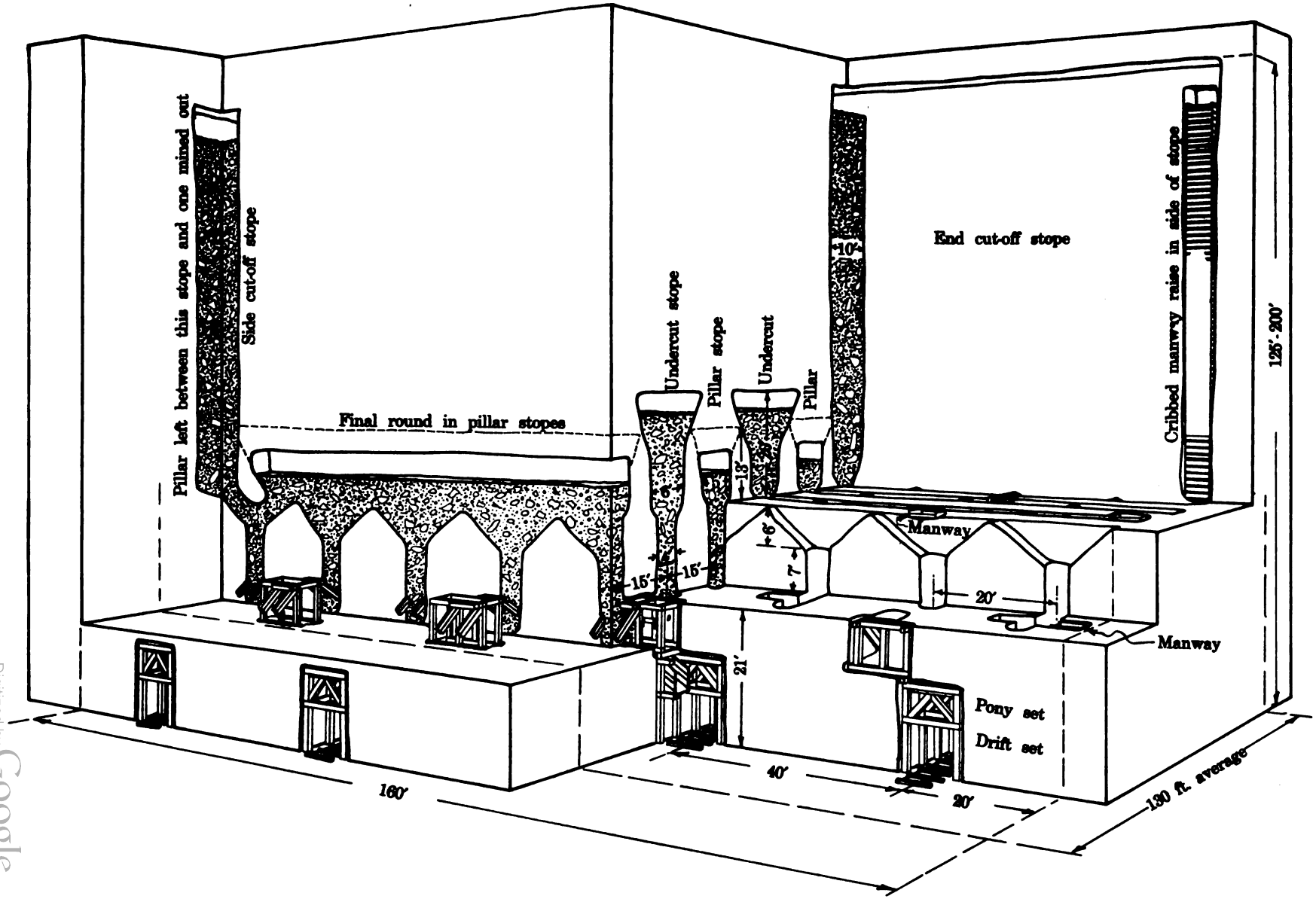

Figure 9. – Modified motor-haulage caving method

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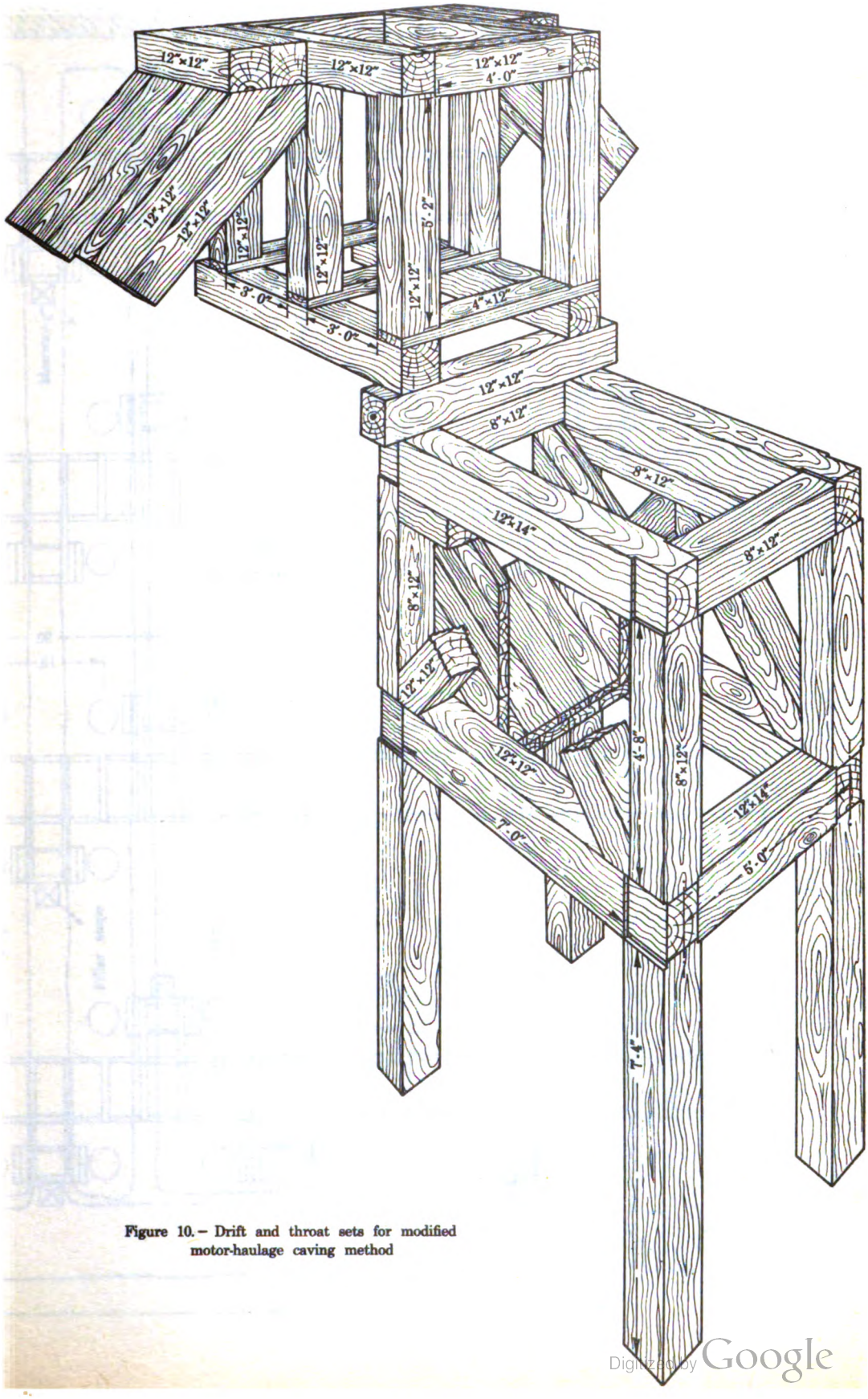

Figure 10. – Drift and throat sets for modified motor-haulage caving method

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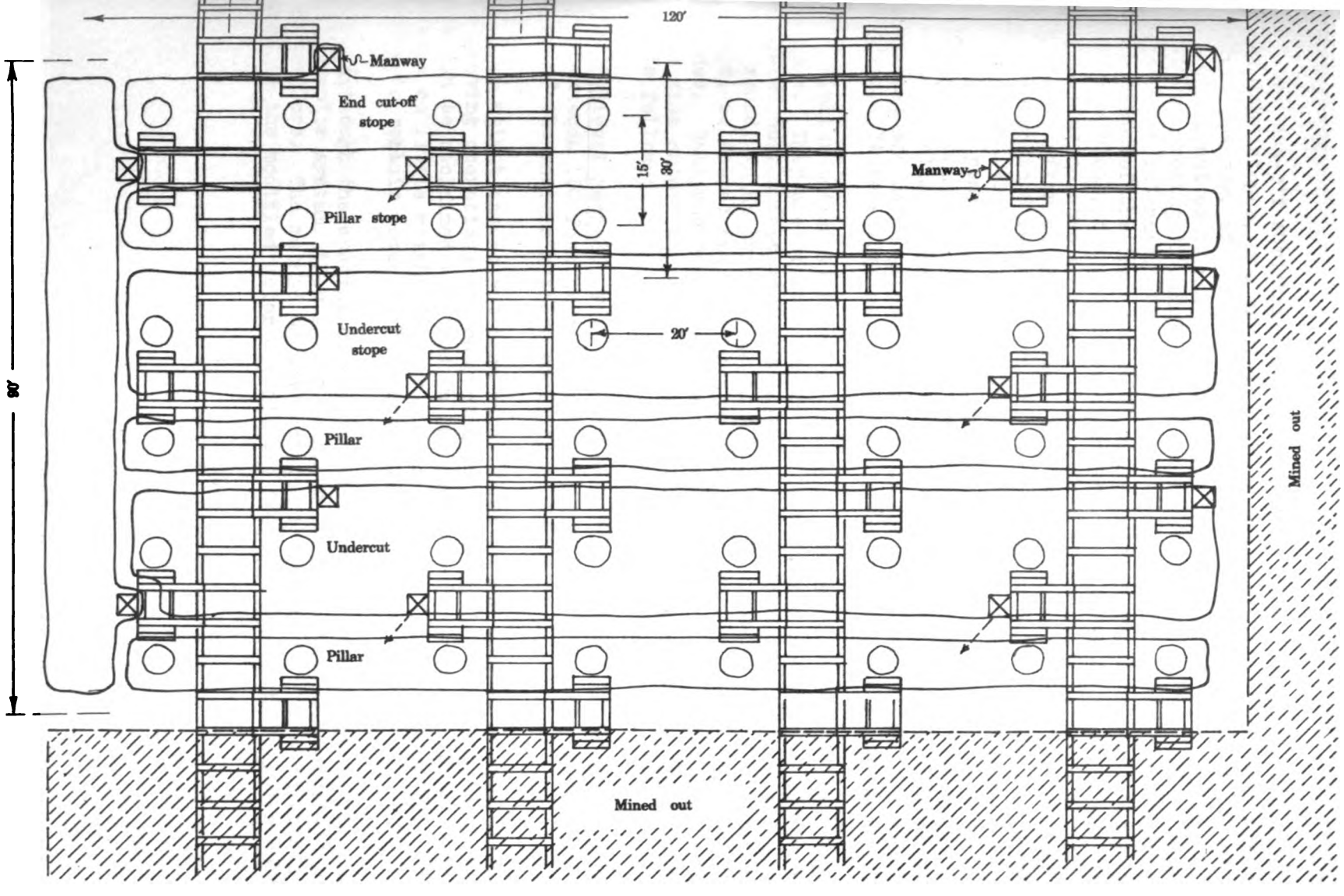

Figure 11.– Plan showing spacing of drifts, throat sets, raises, and stopes in modified motor-haulage caving method

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End cut-off shrinkage stopes are driven at both ends of the block. These, ike the side stopes, are carried to the capping or to a previously mined area above. ıt-off stopes are, of course, unnecessary on such sides of the block as adjoin pre- ıously mined-out sections. Undercutting then begins as in the original motor ethod by connecting 45° inclined raises over the draw points and at right angles) the haulage laterals. These raises are started from the throat raises 7 feet ove the tops of the draw sets. When the raises have been connected across the idth of the block, the ground left between the diverging raises above the throat ises is blasted down to give a slot 4 feet wide with a level back from one end) the other, after which shrinkage stoping commences. The width of the stope is ıen increased to 6 feet for two or three rounds and then again increased so that ıe final round will break a width of at least 18 feet at a height of 40 feet above ıe top of the pony caps. Cribbed manways are put up at the ends and sometimes in ıe center of each undercut stope, starting from the ends of the throat sets. ive feet of solid ground is usually left unbroken between the ends of these trans- erse stopes and caved ground in an adjoining panel. This prevents running in of aste rock. It necessitates, however, careful mapping and plotting of all stopes.

Similar but smaller stopes are next driven in the pillars between the under- ıt stopes. They are started in similar manner by connecting inclined raises, but re carried only 5 feet wide and just one round above the tops of the cones; then final round, which brings them to a height of about 32 feet above the top of the ony caps, is fanned out in such a manner as to break into the undercut stopes on oth sides. Both undercut stopes and pillar stopes connect with their final round ith the side cutoff stope at either end. From this point on, regular drawing pro- edure is followed.

Sublevel Method. The sublevel method is used in all kinds of ground where he ore extends 150 feet or more above the haulage level. It is the most satis- actory of the various methods employed at Ray.

No weight has as yet been experienced in the motor-haulage laterals, and ıfter drawing operations are completed in any area the timbers used in the haulage lrifts can be reclaimed and used in the development of another block. In some ınstances so little weight is encountered on the grizzly, or sublevel, that it is ıossible to reclaim some of this timber also.

Although the spacing of the chutes is not as regular as in the case of the ıodified motor system, they are so arranged as to prevent the formation of ridges of undrawn ore. The method of undercutting the block is almost identical with that pursued in the modified motor method, but the position of the drawing chutes and

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of the grizzly drifts is such that they receive the protection of the cones, or pyramids, of solid rock that are developed in the undercutting procedure. This results in providing for the working places a maximum of protection from the weight of the caving ground with a consequent minimum of expense for maintenance and repairs. A panel 200 feet wide is usually developed by four motor-haulage drifts at 50-foot centers (see fig. 12). Pony sets are installed over every fourth and fifth drift set. One of these has a chute on both sides of the drift, the other is for the convenience and safety of the loader on the motor-haulage level. Raises 4 feet in diameter are driven from these chutes to a height of 40 feet above the rail. The raises are inclined in such a manner as to connect with the sublevel draw laterals at 25-foot centers, and these connections are staggered in alternate sublevel laterals as shown in Figures 12 and 12A.

As soon as this work has progressed far enough to give the required connections to the sublevel, the development of the sublevel is started. This level consists of small timbered fringe drifts on the boundaries of the stoping block and small untimbered drifts or "laterals" across each line of raises from the haulage level. These sublevel laterals are 25 feet center to center and at right angles to the haulage drifts. Over the top of each storage raise a grizzly set with bars 13 inches apart is installed (see fig. 13). After development work has been completed, the undercutting of the block proceeds as in the modified motor method just described. Due to the greater height of the ore, the cut-off stopes, as indicated in Figure 12, are generally mined in two lifts, the upper section of the cut-off stopes being carried up first from an upper level and the shrinkage muck hand-trammed to transfer raises. Figure 14 shows diagrammatically the draw point spacing obtained by the different methods described above.

All drilling in cut-off, undercut, and pillar stopes is done with hand-rotated stoper drills. The back of the stope is kept level lengthwise and slightly arch in section. Drill rounds consist of rows of four holes 6-1/2 feet deep at intervals of 3 or 4 feet along the stope. The final round in undercut stopes and pillar stopes consists of six holes to a row with the outside holes sloped outward on a 45° pitch. As it is essential that the final round in the pillar stopes shall break through into the undercut stopes on both sides, thus making a complete undercut, such rounds are not loaded and blasted until the holes have been checked and approval given by an engineer charged with such responsibility.

Ordinarily, five to ten men per shift work in a stope, depending upon the length of the stope. A round is drilled and blasted over half the length of the stope on each shift. While the crew on one shift is drilling its round in its half of the stope, enough of the swell is drawn from the other half to make room for the

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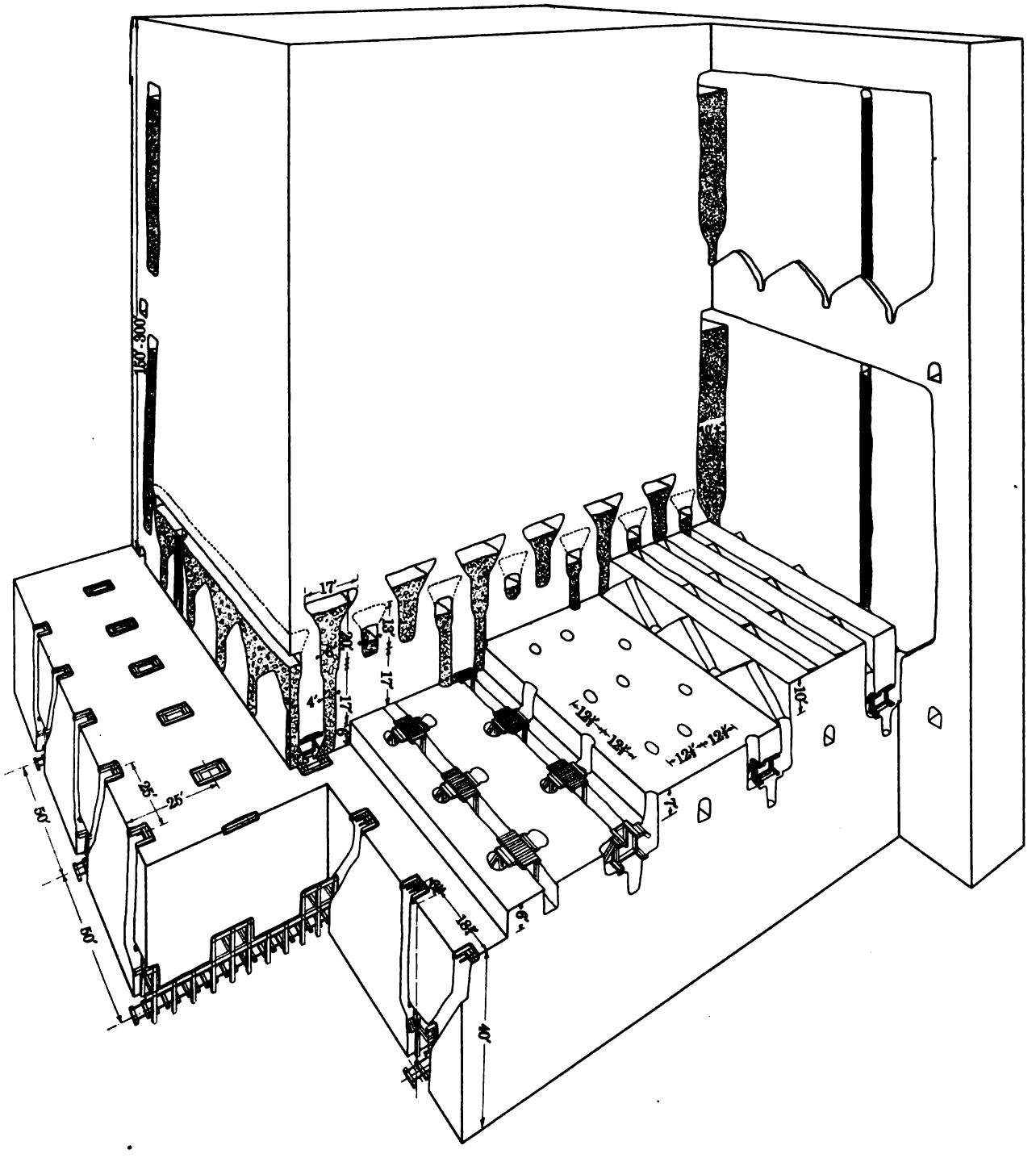

Figure 12 – Sublevel undercut caving method

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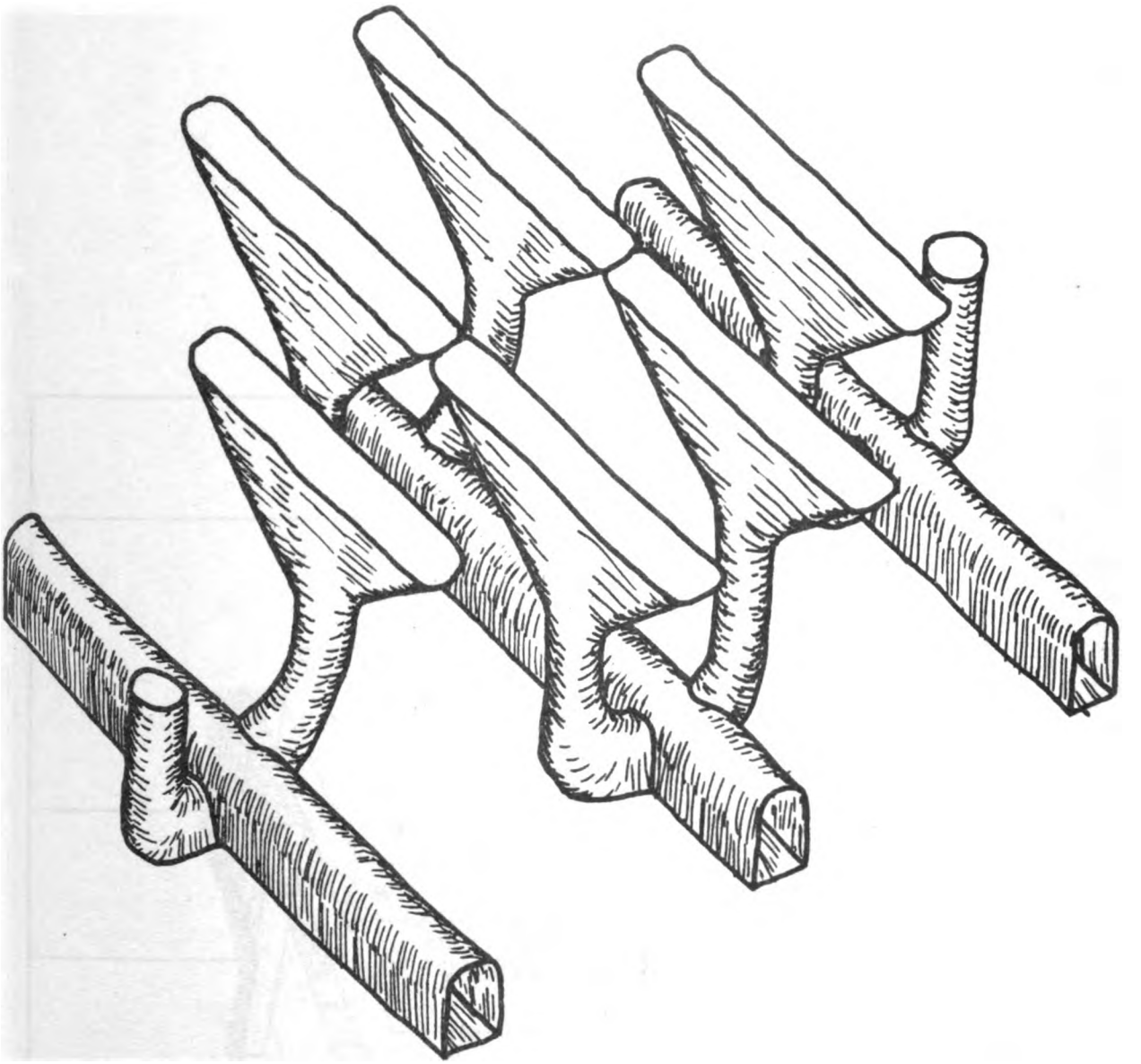

Figure 12A. – Throat raises and slots from sublevel draw laterals up to base of undercutting stopes. Note large amount of solid rock left as protection for laterals

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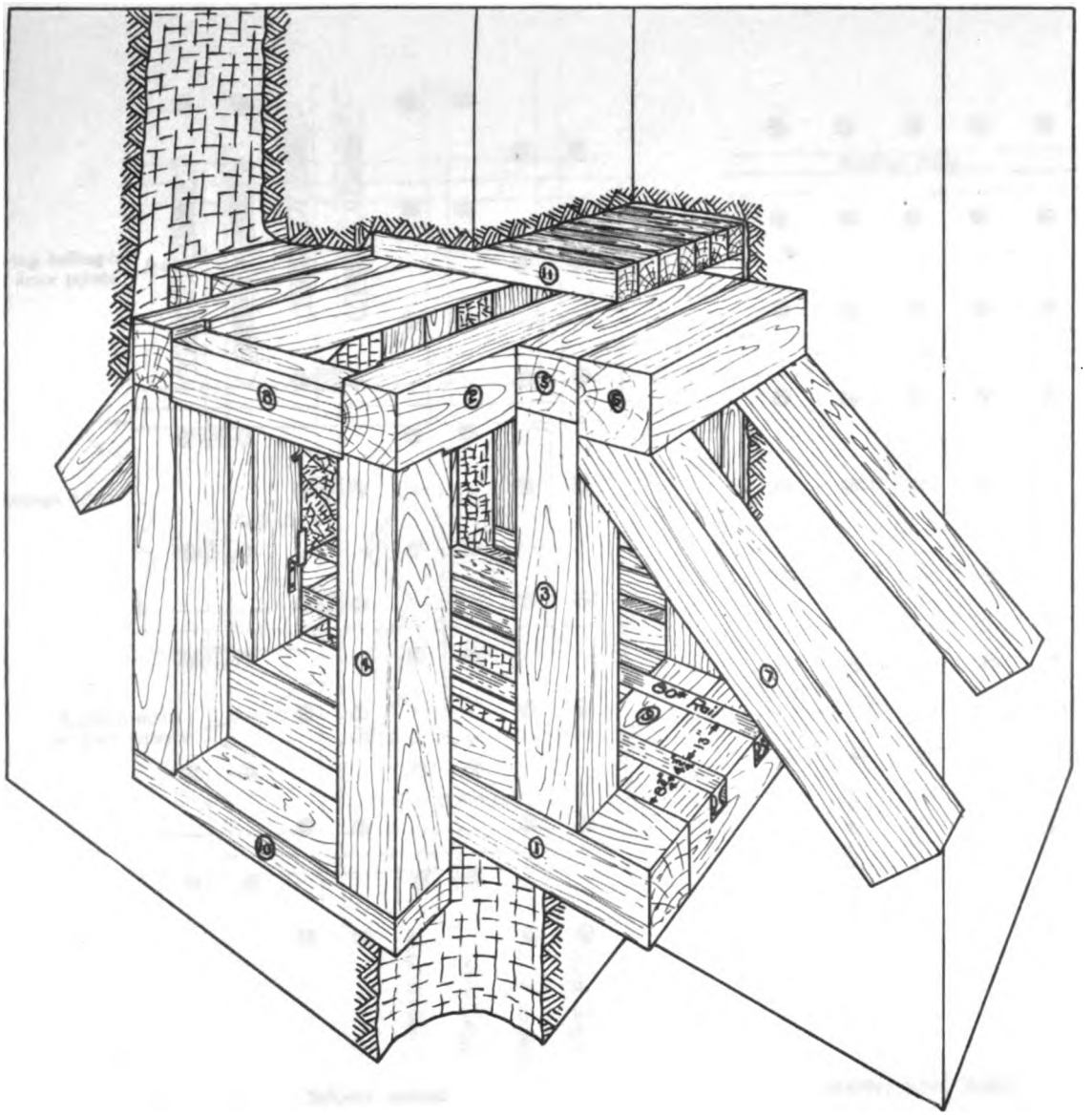

Bill of Material				
№	Pieces	Name	Size	Bd. Ft.
1	2	Stringers	12"×12"×9'-0"	216
2	2	Caps	12×12×10'-0	240
3	4	Posts	12×12×6'-1"	292
4	4	Posts	12×12×8'-1"	388
5	2	Caps	10×12×8'-4	106
6	2	Caps	12×12×6'-4	120
7	4	Posts	12×12×8'-4	240
8	2	Cellar Braces	8×10×8'-8"	48
9	4	Spreaders	12×12×8'-4	160
10	2	Sills	6×12×8'-0	60
11	18	Logging	6×8×8'-0"	300
			Total	1996

Figure 13. – Throat set and grizzly for sublevel caving method

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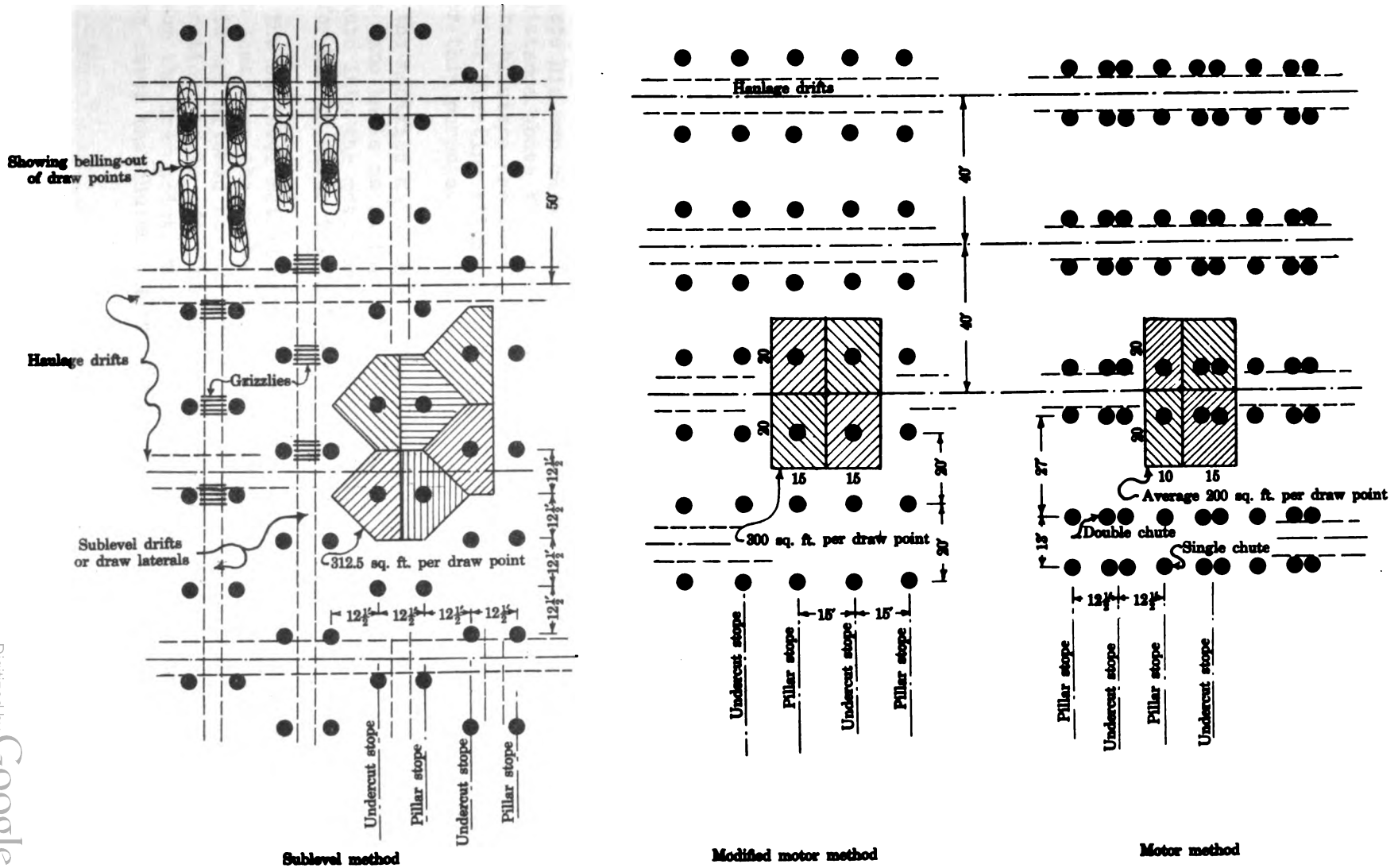

Figure 14. – Diagram of draw point spacings

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drilling of the next round there by the opposite shift. The work in the stope is generally carried out on a two-shift basis; no work is done on the third shift. In each stope a boss has immediate charge of operations under the supervision of the shift boss of that level. Each crew works in its own end of the stope until the stope is finished.

Cribbed stope manways, as previously explained, are put up at the ends and in the middle of the stopes. In the long manways for cut-off stopes, the cribbing is 3 by 6 inch timber and the manway 3-1/2 feet square inside. In the undercut stopes the cribbing is 2 by 6 inch material and the manway 2 feet, 8 inches in the clear. The use of the lighter material in the manways for the undercut stopes is made possible by their temporary nature and the small height to which they are carried.

Much of the ground being mined is soft; where the schistosity lies flat it tends to break into slabs. Under such conditions stulls are frequently used to support the back while drilling is in progress.

Drawing Practices. In hand-tramming sections trammers draw the ore directly from the chutes into 1-ton cars and tram to the transfer raises. One or more chute blasters work in each tramming section, depending on the number of laterals and the nature of the ground being drawn. It is the duty of the chute blaster to blast hung-up chutes, the trammers being prohibited from doing this.

In the motor method the ore is drawn by chute blasters, as they are called, directly from the chutes into the 5-ton motor cars. A train of empties is spotted in the drift and the chute blaster is supplied with empty cars by a "pusher" who brings up the empties and pushes away the loaded cars. In this method the chute blaster blasts his own chutes when they are hung up. At times the rock hangs up at such a distance above the chutes that it is impossible to reach them without a stick. Chute blasters are forbidden to enter hung-up chutes. These are blasted dow with the aid of blasting sticks 1 inch square and from 10 to 15 feet long which are provided for this purpose.

In the modified motor method the drawing operations are conducted on practically the same basis as in the motor section, the chute blaster drawing directly from the chute into the motor-haulage cars. In case the chute hangs up, it is necessary for him to enter the draw set and blast it down.

In the sublevel method drawing is carried on by men designated as tappers to distinguish them from the chute blasters mentioned above. The ore is drawn from the draw sets and allowed to run directly into the storage raises. As in the other drawing operations it is often necessary to blast; and if the muck is hung up any distance from the draw point, this is done with blasting sticks, the tappers being forbidden to enter the chutes. Most of this blasting is done with fuse and cap;

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however, when chutes are hung up in such a manner as to require time and care to place the charge in the correct position, electric blasting caps are used. Thirty per cent dynamite is used in blasting chutes. No blasting is permitted on the grizzlies, hammers being used to break such bowlders as are too large to pass. In order to prevent accidents to the men drawing the storage raises on the haulage level, an electric signal system is used to give warning of blasting to be done on the sublevel.

The method of undercutting used at Ray has been adopted for two reasons. First, to provide pyramids or cones of solid rock above the drawing level which tend to support the broken ground and keep the direct weight from coming upon the draw sets or upon the drawing level. Second, this method of undercutting breaks approximately 45 feet of ground over the draw point. This considerable thickness of broken ore not only acts as a cushion, but as it is drawn off, allows more thorough breaking up of the main mass of ore before the latter reaches the draw points.

A uniform draw, particularly during the early stages of drawing, is essential for satisfactory recovery. If the early drawing is not done uniformly, the mass of ore may cave along some unknown plane of weakness, causing chimneys or cavities which may extend into the capping and allow it to mix with the ore. After the main mass of ore has been thoroughly broken and the block has caved to the surface, the result of subsequent irregular drawing is not as serious as would have been the case earlier. However it is thought best to maintain uniform drawing at all times.

Individual blocks are drawn separately, as each block is cut off on all sides by cut-off stopes. Undercutting operations are carried on progressively from one end of the block to the other, and are immediately followed by drawing operations on a limited scale. This gradually removes the support from under the main mass of ore, and by setting up a cantilever action, causes it to slough and break up. As soon as the block is completely undercut the regular procedure of uniform drawing is started through the block. The same procedure is followed in the next block of the panel, undercutting and drawing being started at the end remote from the previously mined block.

Control of Drawing. Stope control is under the engineering department and is directly supervised by the stope engineer, who lays out stoping plans and calculates the tonnages and assay values of blocks. Under him are stope-control engineers, one on each shift in every working block. It is their duty to see that an even draw is maintained in the drawing section, to take grab samples of the ore at the draw sets, to tally cars on daily written reports, and to seal draw points on

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ccount of waste or overdrawing. Chutes are sealed by cardboard strips tacked to ne chute boards. In the motor-haulage method the number of cars drawn can be llotted directly to the draw points, as each chute represents one draw point only. n the other methods the engineer notes from which side of a draw set the ore is aken, and as the working places are open for his personal observation he is able o keep an accurate check of the tonnage drawn from each individual chute.

Composite assays for each drawing lateral for each shift are posted daily t the mine and at the engineering office. These are for the guidance of the stope ngineer as well as the mine bosses. Individual draw point assays are also posted. n each block a chart shows the number of cars drawn to date from each draw point nd is brought up to date at least twice each week. This serves the foreman and osses by indicating the condition of the draw. A similar chart in the engineering ffice shows cars drawn to date from each draw point, and in addition shows the ori- inal expectancy for each draw point.

Recovery and Dilution. Every effort is made to secure as high a recovery as ossible of the tonnage and copper contents of each individual block. It is impossble in any caving method to obtain 100 per cent recovery of the copper contents, lthough usually the tonnage drawn from any block approximates 100 per cent of its stimated volume. In the final stages of the draw some small percentage of waste aterial is drawn out with the ore and makes up for the small tonnage of commercial re that is left behind in the drawn area. In order that the efficiency of the mining performance in each individual block can be definitely ascertained, it is necesary to estimate as accurately as possible the tonnage and grade of the ore produced rom the block. These data have been kept since the inception of mining operations t Ray, and it is a consideration of these results that has led to the successive mprovements in mining practice that have been instituted at the property. Since he beginning of operations the tonnage mined is somewhat in excess of 40,000,000 ons from completed sections, and the dilution experienced to date approximates 10 er cent, which indicates a reasonably satisfactory mining performance.

Tonnage recovery is the ratio of the tonnage drawn from a block (extraction) o the tonnage estimated as being originally available in that block (expectancy).

Copper recovery is the ratio of pounds of copper drawn to pounds of copper xpected.

Dilution is the loss in grade, and may be calculated by dividing the loss in grade by the expectancy grade of the block.

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Underground Transportation

Figure 2 shows the layout of a typical haulage level and a section through one of the mines. The standard ore car used in this mine has a capacity of 105 cubic feet. The average load is 4.6 dry tons. Haulage is by means of 8-ton compressed-air locomotives with a draw-bar pull of 3,000 pounds. The locomotives are charged at conveniently located stations to a pressure of 900 pounds per square inch the operation requiring about one-half minute. The track is of 35-pound rails in the laterals and 45-pound rails in the main haulage lines, laid to 30-inch gage. Grades of one-third per cent in favor of the load are maintained in the stoping areas and a grade of one-fourth per cent in the main haulage line. Track ties are of untreated 6 by 8 inch timber, 4 feet 6 inches long. Fifty and sixty foot radius curves are standard, though curves of greater radius are frequently used. A standard turnout is shown in Figure 15. Sixteen-inch I-beams are used for the long cars.

Trains consist of 20 to 25 cars. They are brought from the stopes to the main haulage line by the service locomotives and from these haulage to the tipple by the main-line motor. The length of this haul varies greatly but averages approximately 3,000 feet.

The train crew on the main line consists of a motorman and a brakeman. In the sublevel blocks the crew consists of a motorman, a brakeman, and a chute blaster for each train. The chute blaster loads the cars. The brakeman stands below in the drift and signals the motorman in spotting cars. At the tipple the train is turned over to a tipple motorman and his brakeman and an empty train picked up and taken back to the drawing sections.

Motor chutes are of timber construction with chute boards held in cleats.

Cars are dumped by three-car electrically driven tipples. One man operates a tipple. The approximate time of dumping a train of 20 cars is 10 minutes. The ore passes through manganese steel grizzlies, having a spacing of 16 or 18 inches. The capacity of a typical shaft pocket is 570 tons. Stations and pockets are entirely concreted.

Ore is loaded into the skips through measuring hoppers. Skips are of 450-cubic-foot capacity; the average load is approximately 10-1/2 dry tons. At the collar of the shaft the skips dump into a 100-ton steel bin over the coarse crushing plant.

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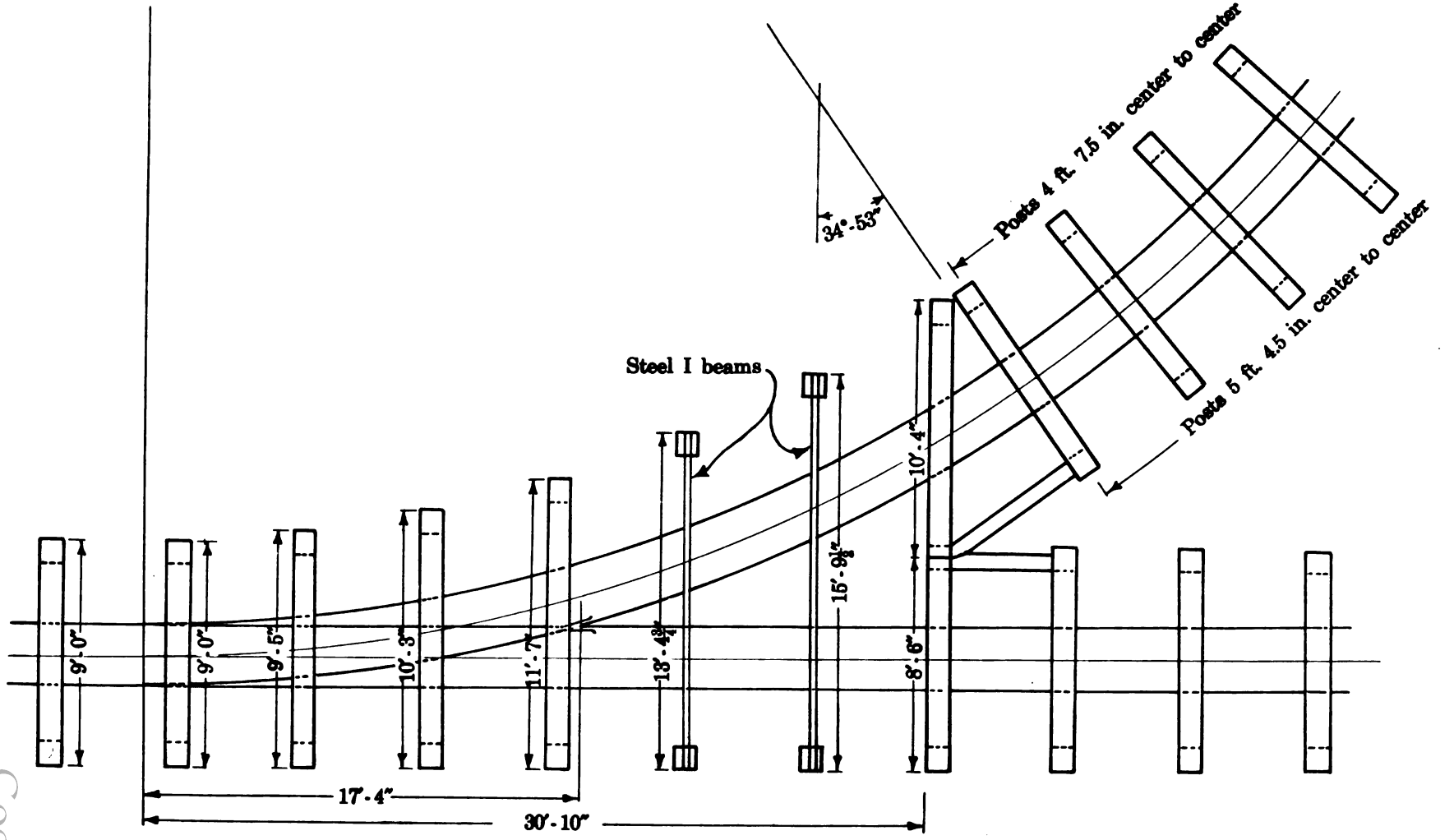

NOTE: All posts on this side of drift are 5 ft. 0 in. center to center

Figure 15. – Timbering of standard 60-foot radius turnout and curve

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METHODS OF PAYMENT

Contracts

A set of standard contract prices is in effect throughout the mine embracing all work that can be handled satisfactorily by the contract system. These prices cover drifting, raising, sinking, timbering, retimbering, undercutting and stoping, and the reclaiming of timber and track material. All prices are on a per foot basis with the exception of stoping, timbering, and the reclaiming of timber and track material. In the case of stoping and undercutting, the price is based on what is locally termed "vertical lineal feet"; that is, a base price is made for each vertical foot of stoping, the actual price for any individual stope being the vertical price per foot times the lineal length of the stope. In case of timbering, a price is given for each set. In reclaiming timber a price is paid on a basis of board feet of timber recovered and in the case of track, pipe, etc., the price is based on the actual lineal feet recovered. The contract rates in effect are seldom altered and then only with the approval of the mine superintendent. Contract work is entirely in charge of the engineering department and is supervised by a contract engineer, whose duty it is to sign new contracts and to keep in touch with the work as it progresses during the month. Contracts are signed for a month's duration only. They are measured on the last day of each month, no excess earnings being paid to the contractors on the mid-month payday. All men on contract are guaranteed day's pay. The contractors themselves are directly responsible to the mine boss in whose section they may be working, and it is the mine bosses' duty to see that the work in carried on in a workmanlike manner. Such earnings as may be made in excess of the daily wage are proportioned to the men working on the basis of their daily wage except that the contractor himself is allowed 25 cents extra per shift for assuming the responsibility of the contract. All contracts cover labor only, and all supplies are furnished by the company. Contracts have been tried at various times to cover the drawing of muck from the chutes, all tappers in any one area being considered on the contract and the basis of pay being so much per car of ore produced. Such contracting improves the efficiency of the drawing operations, but is not satisfactory as it is almost impossible to maintain control of the draw.

Bonus

With the present contract system, which borders on the bonus system, very little actual bonus work is done. A bonus is paid trammers for the tramming of muck from chutes, and a definite amount per car is paid for all cars in excess of a certain number. The base number of cars depends on the length of tram. In addition to the contract price paid for drifting and raising work, a bonus is sometimes allowed on rush headings of so much per foot, providing a certain footage is completed during the month.

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A bonus is also applied to work in which great difficulty is being experienced. This in effect raises the contract price, except that the bonus is not paid unless some agreed footage is completed during the period.

VENTILATION

The principal ventilation requirement is the rapid removal of blasting gases, smoke, and dust. The No. 2 shaft workings are mechanically ventilated by three nonreversible exhaust fans. The major part of these workings is ventilated by a centrifugal fan installed at the collar of a ventilating shaft, exhausting 75,000 cubic feet per minute under present conditions. This air intakes through one ventilation shaft and one inclined supply shaft and, to a small extent, through surface cracks. It ventilates six stoping areas on two circuits.

A smaller and somewhat isolated section of the mine embracing two active stoping areas is ventilated by a similar fan installed at the portal of a 500-foot crosscut leading to another ventilation shaft. Under present conditions it exhausts 40,000 cubic feet per minute of air that intakes through the No. 2 incline shaft. This system is so arranged that it can also ventilate the No. 1 mine.

The three main haulage level tipples are separately ventilated by air intaking through No. 2 main shaft and No. 2 incline, and exhausting through a vent shaft by means of a similar type of fan installed at the collar of No. 2 incline. It exhausts 42,000 cubic feet of air per minute under present conditions.

The man shaft is closed off and nominally neutral.

Sublevels are ventilated by deflecting the air current to and from them through raises by means of doors in the main-haulage level laterals. Stopes are ventilated by drill exhaust and circulation through manway raises, also through open overhead raises driven primarily for ventilation.

Air samples show that the quality of the air, as to chemical composition, is good throughout the mine. The total air supply is equivalent to 400 cubic feet per minute per man employed underground on the day shift.

MINE DRAINAGE

The amount of water encountered in these mines is small. At the No. 1 shaft approximately 160 gallons per minute is being pumped, and about 40 gallons per minute at the No. 2 shaft. This water is handled by electrically driven triplex pumps.

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SAFETY METHODS, FIRST-AID ORGANIZATION AND TRAINING, AND FIRE PREVENTION

The prevention of accidents has always been considered an important phase of the operations at Ray. For this reason an engineer trained in mine-safety work is a member of the operating staff. His duties are: (1) To be familiar with operations throughout the plant, with a view to decreasing accident hazards; (2) to carry on educational work to the same end; (3) to give instruction in first-aid and mine-rescue work; (4) to keep records of accidents which will direct attention to the important points of attack in the work of his department.

Accident Prevention

Operating regulations have been formulated covering all phases of the work, both in the mines and on the surface. These rules, written in both English and Spanish, have been posted on bulletin boards in various places.

Cash awards are made to the bosses each month, first for a perfect record, second for the least number of accidents per shift worked, and third for the least number of days lost per 1000 shifts worked. These monthly awards are graduated and are based on the number of man-shifts worked per month. As a result of this policy all bosses have come to regard the elimination of careless or incompetent workmen from their crews and the building up of an accident-proof organization as one of their most important duties.

Accident-prevention education is fostered by exhibits, signs, bulletin boards, and weekly safety-first letters. Voluntary meetings of particular groups of workmen are held at which the work of each group is discussed and safe practices taught. The supervisory force meets at luncheons each month for the discussion of accident prevention.

First-Aid and Mine-Rescue Work

The safety engineer gives instruction in first aid. All bosses are required to take the course and it is estimated that one-third of all men on the pay roll are competent to administer first aid in case of emergency. Supplies of dressings and bandages are kept in the mine in charge of the shift bosses. Completely equipped first-aid stations are maintained at centers of work throughout the mines and plant.

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Five mine-rescue crews composed of six men each are kept in training. For this purpose a mine-rescue station has been built and equipped with all necessary apparatus, such as breathing apparatus, gas masks, oxygen pumps, and supplies.

Mine-Fire Prevention

The danger of fire in the main shafts is eliminated by concreting, as before mentioned. Fire trucks are kept on each level, and the mine is equipped throughout with fire hydrants, exit signs, and lights. Special sets of fire and ventilation maps are kept up-to-date, and plans are prepared for fighting any possible fire.

ORGANIZATION

Figure 16 is a chart of the administrative organization of the Ray Mines. It should be noted that in the absence of the mine superintendent the assistant mine superintendent assumes the authority of the former.

Table 1.--Production of ore by each variation of the mining method and by preparatory excavation during the year 1928

	Tons	Percentage of total
Preparatory excavation	186,742	5.76
Hand-tramming method	119,981	3.70
Regular motor method	388,154	11.97
Modified motor method	294,769	9.09
Sublevel method	2,253,513	69.48
Total mine production	3,243,159	100.00

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Table 2.--Summary of costs, year 1928

	Labor	Timber	Explosives	Power	Machine shop	Misc.	Total
Preparatory excavation	$0.0581	$0.0339	$0.0070	$0.0014	$0.0010	$0.0022	$0.1036
Stoping	.0711	.0082	.0247	.0050	.0016	.0059	.1165
Reserve drawing	.0940	.0129	.0190	.0001	.0001	.0006	.1267
Haulage (including operation tipples)	.0585	.0063	--	.0078	.0139	.0101	.0909
Hoisting	.0084	.0001	--	.0071	.0014	.0027	.0197
Misc. mine expense	.0368	.0001	--	.0114	.0016	.0074	.0573
Plant administration and general	.0445	.0005	--	.0023	.0018	.0602	.1093
General overhead (exclusive of taxes)	--	--	--	--	--	.0526	.0526
Totals	0.3714	0.0563	0.0507	0.0351	0.0214	0.1417	0.6766

Table 3.--Labor, powder, and timber consumption

Year	Tons mined	Tons per man			Pounds powder per ton	Board feet per ton
		Stoping	Mine pay roll	Over-all		
1927	2,551,913	101.60	10.66	9.39	.2523	1.79
1928	3,243,159	1/115.61	12.13	10.83	.3193	1.88

1/ This figure estimated because of change in cost charges between the years 1927 and 1928.

Table 4.--Labor, powder, and timber consumption--development, 1928

	Size of heading, feet	Feet per man-shift	Pounds powder per foot	Board feet timber per foot
Grizzly drifts (untimbered)	4 by 5	2.16	4.84	--
Supply drifts (timbered)	6 by 8	1.16	5.39	50
Motor drifts (timbered)	8-1/2 by 9	.84	6.92	113
Averages total driving and timbering	--	1.05	--	--
Throat raises	4 ft. dia.	9.2	4.17	--
Short haulage raises (untimbered)	4 by 4	5.67	4.72	--
Cribbed raises	5 by 5	2.58	4.90	80
Average raises	--	4.50	--	--
Grand averages, all driving and timbering	--	1.38	5.69	52.38

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Table 5.--Estimated cost of preparatory work for a block containing 560,000 tons[1], Ray Mines, Nevada Consolidated Copper Co.

	Tons	Shifts	Units	Cost per unit				Total cost, labor timber, and explosives	
				Labor	Timber	Explosive	Total	Amount	Cost per ton[2]
Supply drifts	1,625	377	325 ft.	$ 3.94	$1.93	$0.74	$6.61	2,148.25	
Grizzly drifts	4,805	674	1,456 ft.	1.83	-	0.66	2.54	3,698.24	
Grizzlies	1,310	784	56	82.00	81.83	4.52	168.35	9,427.60	
Totals	7,740	1,835	1,781 ft.	-	-	-	-	15,274.09	$ .027
Shrinkage stopes[3]	16,240)	-	(16,240 tons)						
	)	4,236	( )	0.63	0.11	0.32	1.06	29,124.56	
Undercutting[3]	11,236)	-	(11,236 tons)						
Totals	27,476	4,236	27,476 tons	-	-	-	-	29,124.56	.052
Total above grizzly level	35,216	6,071	-	-	-	-	-	44,398.65	.079
Haulage drifts	6,090	833	700 ft.	6.53	4.37	0.95	11.85	8,295.00	
Pony sets and chutes	2,800	130	560 ft.	3.85	3.82	0.22	7.89	4,418.40	
Haulage raises	5,544	326	1,848 ft.	.31	-	0.65	1.56	2,882.88	
Total below grizzly level	14,434	1,289	3,108 ft.	-	-	-	-	15,596.28	.028
Total development exclusive of shrinks and undercuts	22,174	3,124	-	-	-	-	-	30,870.37	.055
Grand total	49,650	7,360	-	-	-	-	-	59,994.93	.107

1/ - Area = 35,000 square feet; lift = 200 feet. 2/ Costs per ton are based on 560,000 tons production. 3/ Tonnage used is swell only.

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Table 5.--Estimated cost of preparatory work for a block containing 560,000 tons[1], Ray Mines, Nevada Consolidated Copper Co. (Continued)

	Haulage and hoisting		Assaying, ventilation, and sampling		Machine drills		Miscellaneous mine expenses		Grand total	
	Amount	Cost per ton[1]	Amount	Cost per ton[1]	Amount	Cost per ton[1]	Amount	Cost per ton[1]	Amount	Cost per ton[1]
Supply drifts	167.37		37.38		287.27		55.25			
Grizzly drifts	494.92		110.52		849.43		163.37			
Grizzlies	134.95		30.12		231.60		44.54			
Totals	797.24	.001	178.02	.0003	1,368.30	.0024	263.16	.0005	17,380.81	.0312
Shrinkage stopes[2] ....) Undercutting[2] .....)	2,830.01	-	631.95	-	10,080.00	-	934.18			
Totals	2,830.01	.005	631.95	.0011	10,080.00	.018	934.18	.0016	43,600.70	.0777
Total above grizzly level	3,627.25	.006	809.97	.0014	11,448.30	.0204	1,197.34	.0021	61,481.51	.1089
Haulage drifts	627.27		140.07		1,076.65		207.06			
Pony sets and chutes .	288.40		64.40		494.98		95.20			
Haulage raises	571.03		127.51		980.07		188.50			
Total below grizzly level	1,486.70	.003	331.98	.0006	2,551.70	.0046	490.76	.0009	20,457.42	.0371
Total development exclusive of shrinks and undercuts	2,283.94	.004	510.00	.0009	3,920.00	.007	753.92	.0014	38,338.23	.0683
Grand total	5,113.95	.009	1,141.95	.002	14,000.00	.025	1,688.10	.003	81,938.93	.146

1/ Costs per ton are based on 560,000 tons production - 2/ Tonnage used is swell only.

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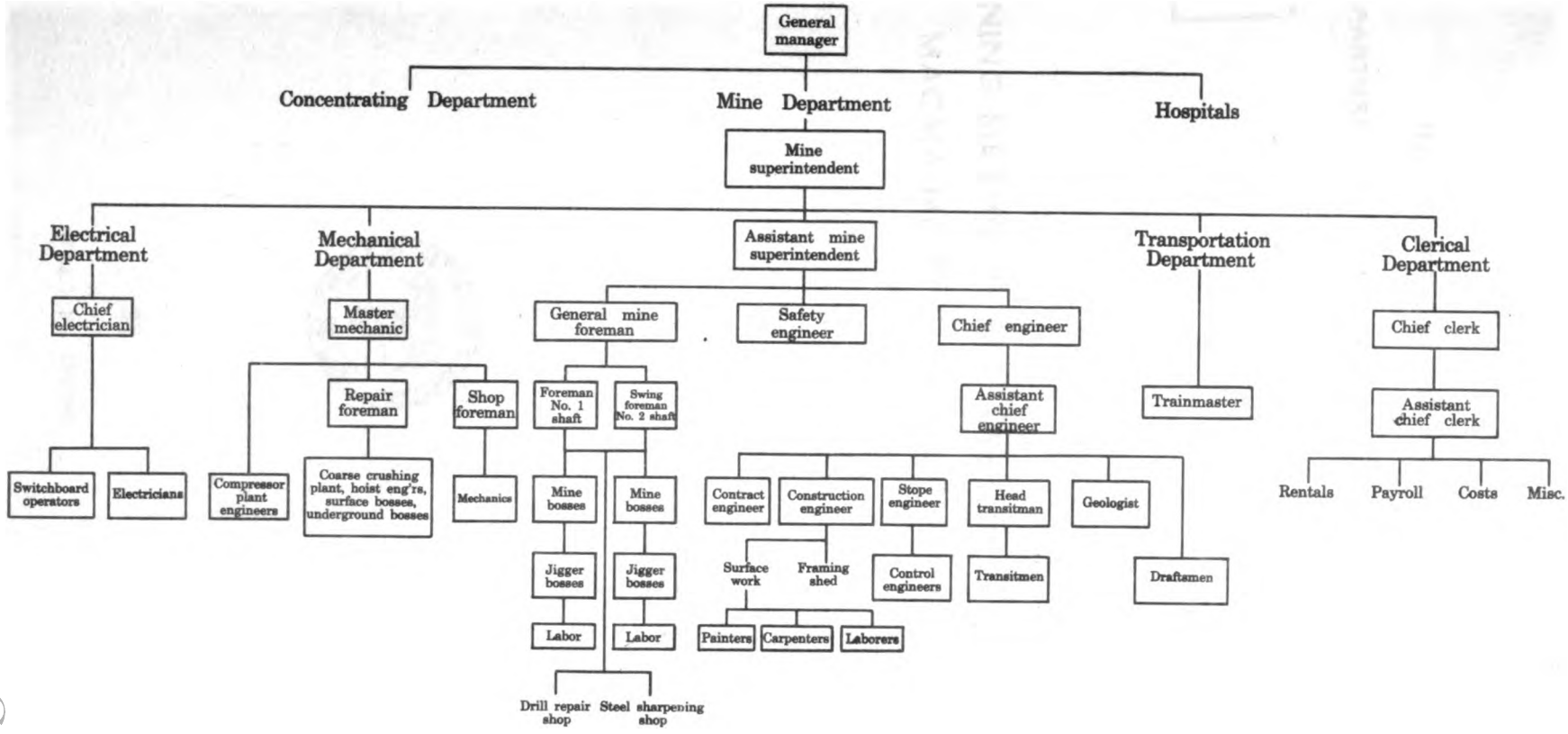

Figure 16 – Organization chart

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